Lake Superior Mining Institute

PROCEEDINGS—1912

Lake Superior Mining Inst.
VHA

PROCEEDINGS

OF THE

LAKE SUPERIOR MINING INSTITUTE

SEVENTEENTH ANNUAL MEETING

HOUGHTON, MICHIGAN

AUGUST 28, 29, 30, 1912

VOL. XVII.

ISHPEMING, MICH.
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AT THE OFFICE OF THE SECRETARY
1912

PRESSES OF IRON ORE
ISHPEMING, MICH.

INDEX TO VOLUMN XVII.

PAPERS.

ERRATA.

Page 141, Line 11—Strike out "with"; after "feet" read "at first, then salt with rock."

Page 142, Line 1—After "report," read "Monograph."

8—For "as" read "is."

5—(From bottom) After "beds" read "are continuous."

Page 231, Line 26—"Final set" should read "final act."

Page 232, Line 2—Cancel fourth word "of."

22—"Per se" italics.

OFFICERS.

For the Year Ending With the Close of the Annual Meeting August 30th, 1912.

PRESIDENT.

F. W. DENTON .... Painesdale, Mich.
(Term one year).

VICE PRESIDENTS.

E. D. BRIGHAM .... Chicago, Ills.
C. H. MUNGER .... Duluth, Minn.
(Term expires 1912).

GEO. H. ABEEL .... Ironwood, Mich.
GRAHAM POPE .... Houghton, Mich.
W. H. JOBE .... Palatka, Mich.
(Term expires 1913).

MANAGERS.

PETER PASCOE .... Republic, Mich.
J. B. COOPER .... Hubbell, Mich.
L. C. BREWER .... Ironwood, Mich.
(Term expires 1912).

M. H. GODFREY .... Coleraine, Minn.
J. E. JOPLING .... Ishpeming, Mich.
(Term expires 1913).

TREASURER.

E. W. HOPKINS .... Commonwealth, Wis.
(Term expires 1912).

SECRETARY.

A. J. YUNGBLUTH .... Ishpeming, Mich.
(Term expires 1912).

OFFICERS.

The following is list of officers elected at the annual meeting, August 30th, 1912, also the officers holding over from the previous year and which are indicated by *.

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PENTECOST MITCHELL .... Duluth, Minn.
(Term one year).

VICE PRESIDENTS.

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†W. P. CHINN .... McKinley, Minn.
*W. H. JOBE .... Palatka, Mich.
(Term expires 1913).

FRANCIS J. WEBB .... Duluth, Minn.
A. D. EDWARDS .... Atlantic Mine, Mich.
(Term expires 1914).

MANAGERS.

*M. H. GODFREY .... Coleraine, Minn.
*JAMES E. JOPLING .... Ishpeming, Mich.
(Term expires 1913).

G. S. BARBER .... Bessemer, Mich.
WM. H. JOHNSTON .... Ishpeming, Mich.
C. H. BAXTER .... Loretto, Mich.
(Term expires 1914).

TREASURER.

E. W. HOPKINS .... Commonwealth, Wis.
(Term expires 1913).

SECRETARY.

A. J. YUNGBLUTH .... Ishpeming, Mich.
(Term expires 1913).

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†To fill vacancy of Graham Pope, deceased.

LIST OF STANDING COMMITTEES FOR YEAR ENDING 1913.

PRACTICE FOR THE PREVENTION OF ACCIDENTS.

J. E. JOPLING, Chairman ....Ishpeming, Mich.
CHAS. E. LAWRENCE ....Iron Mountain, Mich.
D. E. SUTHERLAND ....Ironwood, Mich.
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EDWARD KOEPEL ....Beacon Hill, Mich.

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J. S. JACKA ....Crystal Falls, Mich.
WM. A. COLE ....Ironwood, Mich.
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PAPERS AND PUBLICATIONS.

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WM. H. JOHNSTON ....Ishpeming, Mich.
J. M. BUSH ....Ironwood, Mich.
J. H. HEARDING ....Duluth, Minn.
F. W. McNAIR ....Houghton, Mich.

BUREAU OF MINES.

M. M. DUNCAN, Chairman ....Ishpeming, Mich.
J. B. COOPER ....Hubbell, Mich.
A. J. YUNGBLUTH, Secretary ....Ishpeming, Mich.

BIOGRAPHY.

J. H. HEARDING, Chairman ....Duluth, Minn.
W. J. OLCOTT ....Duluth, Minn.
GEORGE A. NEWETT ....Ishpeming, Mich.
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POMPELLY, RAPHAEL ....Dublin, N. H.
VAN HISE, C. R....................................Madison, Wis.
WINCHELL, N. H...........501 East River Road, Minneapolis, Minn.

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ABBOTT, C. E.......................................Bessemer, Ala.
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ADGATE, FREDERICK W..........419 Rookery Bldg., Chicago, Ills.
AISHTON, R. H....................215 Jackson Blvd., Chicago, Ills.
ALLEN, R. C.......................................Lansing, Mich.
ALVER, G. A.......................................Norway, Mich.
AMBERG, J. W.......................1400 Fulton St., Chicago, Ills.
AMBERG, WILLIAM A...............1400 Fulton St., Chicago, Ills.
ANDERSON, G. A...............................Deerwood, Minn.
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ANGST, ROBERT A.................................Hibbing, Minn.
APPLEBY, WILLIAM R........School of Mines, Minneapolis, Minn.
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ATKINS, SAMUEL E..............909 Alworth Bldg., Duluth, Minn.

BAER, HENRY L.................................Hancock, Mich.
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BANDLER, ARTHUR S................65 Nausau St., New York City
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BARBER, MAX H....................................Nashwauk, Minn.
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BOWEN, REUBEN ....Pittsburg, Pa.
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BRIGHAM, E. D...................215 Jackson Blvd., Chicago, Ills.
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BURT, JOHN H..................................Virginia, Minn.
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CARROL, MICHAEL J............................Houghton, Mich.
CARSON, JOHN A..................................Appleton, Wis.
CARTER, RAYMOND B...........301 W. Randolph St., Chicago, Ills.
CHAMBERLAIN, PAUL FAIRFIELD..................Soudan, Minn.
CHAMBERLIN, H. L...........................Michigamme, Mich.
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CHAMPION, JOHN ....Humboldt, Mich.
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CUMMINGS, P. H...............................Ironwood, Mich.

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CURRY, SOLOMAN S....Ironwood, Mich.
CURRY, GEORGE A....Ironwood, Mich.

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HUNTER, ROY D........1506 Railway Exchange Bldg., Chicago, Ills.
HURTER, CHARLES S.............DuPont Bldg., Wilmington, Del.

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IRELAND, JAMES D...............701 Sellwood Bldg., Duluth, Minn.

JACKA, JOSIAH S............................Crystal Falls, Mich.
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JACKSON, GEORGE R...........................Princeton, Mich.
JACKSON, J. F.............695 Colby-Abbot Bldg., Milwaukee, Wis.
JACKSON, FRANK W.......Market and Randolph Sts., Chicago, Ills.
JAEHNIG, FRANK A..............................Dunmon, Mich.
JANSON, F. A....................................Norway, Mich.
JENSON, N. C......................25 Broad St., N., New York City
JETTNER, AUCUST R.............176 Randolph St., Chicago, Ills.
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JONES, CHARLES C................................Ramsey, Mich.
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JONES, JOHN T.............................Iron Mountain, Mich.
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JORY, WILLIAM ....Princeton, Mich.

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KEARNEY, F. H..................................Ironwood, Mich.
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KENNEDY, F. A...................................Hibbing, Minn.

KING, ROBERT ....Hurley, Wis.
KIRKPATRICK, J. CLARK........................Escanaba, Mich.
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KNIGHT, R. C....................................Eveleth, Minn.
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KOENIG, GEORGE A..............................Houghton, Mich.
KOEPEL, ED....................................Beacon Hill, Mich.
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LEACH, EDWARD J................................Hancock, Mich.
LEIN, NELS ....Eveleth, Minn.
LEOPOLD, N. F...................108 Dearborn St., Chicago, Ills.
LETZ, JOHN F........................662 12th St., Milwaukee, Wis.
LIBBY, DR. E. M..............................Iron River, Mich.
LINDBERG, JOHN FREDERICK.......................Hibbing, Minn.
LINN, A. E.......................................Norway, Mich.
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MATTHEWS, WILLIAM C.....................Wilmington, Del.
MEADS, ALEXANDER P.......................Marquette, Mich.
MENNIE, JOHN S.............................Ishpeming, Mich.
MERCER, HARRY T...........................Painesdale, Mich.
MEUCHE, A. H...............................Houghton, Mich.
MEYERS, WILLIAM R..........................Princeton, Mich.
MILLAR, JOHN M.............................Escanaba, Mich.
MILLER, L. B......................Wade Bldg., Cleveland, Ohio
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MITCHELL, R. J..............................Eveleth, Minn.
MITCHELL, WILLIAM A.......16th and Rockwell Sts., Chicago, Ills.
MITCHELL, SAMUEL J........................Marquette, Mich.
MOELLER, FRANKLIN ....42 Chapman Ave., Cleveland, Ohio
MONROE, W. G.........................Iron Mountain, Mich.
MOORE, C. F.............920 Newhouse Bldg., Salt Lake City, Utah
MOORE, CLARENCE E..........................Virginia, Minn.
MORGAN, DAVID T...........................Ishpeming, Mich.
MORRIS, CHARLES S...............2232 E. First St., Duluth, Minn.
MOULTON, W. H.............................Ishpeming, Mich.
MOWATT, NEVILLE P........1432 McCormick Bldg., Chicago, Ills.
MULLEN, THOMAS M..........................Houghton, Mich.
MUNGER, CHARLES H...........................Duluth, Minn.
MUNROE, HENRY S...........Columbia University, New York City
MURPHY, C. M...............................Ishpeming, Mich.
MURRAY, ROBERT ....Hibbing, Minn.
MYERS, ALBERT J.......................Iron Mountain, Mich.
M'CLURE, O. D..............................Ishpeming, Mich.
M'CORMICK, EDWARD ....Hibbing, Minn.
M'CORKINDALE, W. J.........................Ishpeming, Mich.
M'DONALD, D. B..............................Duluth, Minn.
M'DOWELL, JOHN ....Hibbing, Minn.
M'GEE, M. B..............................Crystal Falls, Mich.
M'GONAGLE, WILLIAM A.............Wolvin Bldg., Duluth, Minn.
M'GREGOR, SILAS J......................Iron Mountain, Mich.
M'INDOE, JAMES A............................Norway, Mich.
M'INTYRE, JOHN E...........................Nogales, Arizona
M'KEEVER, R. T.................82 Devonshire St., Boston, Mass.

M'KEE, WILL E....................................Warren, Ariz.
M'LAUGHLIN, W. J..................................Loretto, Mich.
M'LEAN, JOHN H....................................Duluth, Minn.
M'LEAN, RICHARD EARLE.................Wells, Delta Co., Mich.
M'NAMARA, THOMAS B..........................Ironwood, Mich.
M'NAMARA, TIMOTHY ....Fierro, New Mexico
M'NAIR, F. W....................................Houghton, Mich.
M'NEIL, E. D......................................Virginia, Minn.

NELSON, S. T......................1170 W. Lake St., Chicago, Ills.
NEWBY, WILLIAM ....Ironwood, Mich.
NEWETT, GEORGE A..........................Ishpeming, Mich.
NEWETT, W. H..................................Ishpeming, Mich.
NEWTON, L. L..................1324 La Salle Ave., Chicago, Ills.
NICHOLAS, THOMAS J................................Aurora, Minn.
NICHOLS, F. W....................................Houghton, Mich.
NICKERSON, H. F..................................Houghton, Mich.
NIXON, JOHN A....................................Ishpeming, Mich.
NOBLE, THOMAS H..............................Marquette, Mich.
NOETZEL, BENJAMIN D.......................Trimountain, Mich.

OLCOTT, WILLIAM J................................Duluth, Minn.
ORBISON, THOMAS W..............................Appleton, Wis.
ORR, FRANK D......................Exchange Bldg., Duluth, Minn.
OSBORN, CHASE S........................Sault Ste. Marie, Mich.

PAINE, W. A.....................82 Devonshire St., Boston, Mass.
PAINE, FRANCIS W................................Houghton, Mich.
PARKER, RICHARD A..............929 Foster Bldg., Denver, Colo.
PASCOE, PETER W...................................Republic, Mich.
PATRICK, RICHARD S.........314-15 Sellwood Block, Duluth, Minn.
PENGILLY, EDWARD ....Crystal Falls, Mich.
PENTECOST, JOHN ....Calumet, Mich.
PENTON, JOHN A..............Iron Trade Review, Cleveland, Ohio
PERKINS, SAMUEL J..............................Ironwood, Mich.
PHILLIPS, W. G....................................Calumet, Mich.
PITKIN, S. H....................682 W. Market St., Akron, Ohio
PLATTO, FRANK ....Ishpeming, Mich.
POTTER, OCHA ....Houghton, Mich.
POWELL, D. W....................................Marquette, Mich.
PRESCOTT, FRED M...................Oregon St., Milwaukee, Wis.
PRESCOTT, L. L..................................Menominee, Mich.
PRYOR, R. C..Houghton, Mich.

QUIGLEY, G. J......................................Antigo, Wis.
QUINE, JOHN THOMAS.............413 Vine St., Ishpeming, Mich.
QUINN, CLEMENT KRUSE..........................Virginia, Minn.

RAHT, CHARLES ....29 Broadway, New York City
RAISKY, F. H....................................Ishpeming, Mich.
RALEY, ROBERT J..................Spalding Hotel, Duluth, Minn.
RANKIN, WILLIAM A..........................Painesdale, Mich.
RASHLEIGH, WILLIAM J..........................Aurora, Minn.
RAYMOND, HENRY A.............Rockefeller Bldg., Cleveland, Ohio
REDFERN, JOHN A..................................Hibbing, Minn.
REEDER, J. T.....................................Houghton, Mich.
REEDER, EDWIN C................1917 Fisher Bldg., Chicago, Ills.
REEDER, J. H.....................................Houghton, Mich.
REHFUSS, LOUIS I...............................LaCrosse, Wis.
REIGART, JOHN R...............................Princeton, Mich.
REYNOLDS, M. K................430 E. Arch St., Marquette, Mich.
RICE, CLAUDE T..Eng. & Min. World, Monadnock Bldg., Chicago, Ills.
RICE, JOHN H.....................................Houghton, Mich.
RICHARDS, WILLIAM J......................Crystal Falls, Mich.
RICHARDS, MORRIS EARL........................Virginia, Minn.
RICHARDS, WILLIAM J.........................Painesdale, Mich.
RICHEY, E. W............211 Railway Exchange Bldg., Chicago, Ills.
RIDLEY, FREDERICK WILLIAM.....................Calumet, Mich.
ROBERTS, HARRY ....Duluth, Minn.
ROBERTS, RICHARD ....Negaunee, Mich.
ROBERTS, ALTON T..........................Marquette, Mich.
ROBERTSON, JAMES ....Houghton, Mich.
ROBERTSON, HUGH J..........................Escanaba, Mich.
ROBINSON, JAMES A.............................Hibbing, Minn.
ROHN, OSCAR ....Butte, Mont.
ROSE, R. S.....................................Marquette, Mich.
ROSKILLY, JOSEPH ....Virginia, Minn.
ROUGH, JAMES H..............................Negaunee, Mich.
ROWE, HENRY ....Ironwood, Mich.
ROWE, WM. C...................................Bessemer, Mich.
RUEZ, GEORGE F...............................Ishpeming, Mich.
RUMSEY, SPENCER S...............610 Wolvin Bldg., Duluth, Minn.
RUNDLE, A. J..............................Iron Mountain, Mich.
RUSSELL, C. W......................R. W. & F. Co., Detroit, Mich.
RYAN, JOHN A..............................Iron Mountain, Mich.

SALSICH, L. R..................................Coleraine, Minn.
SAVAGE, J. A.....................................Hibbing, Minn.
SCADDEN, FRANK ....Crystal Falls, Mich.
SCHACHT, WILLIAM H.........................Painesdale, Mich.
SCHLESINGER H. J............................Milwaukee, Wis.
SCHUBERT, GEORGE P...........................Hancock, Mich.
SCHULZE, W. J...................................Virginia, Minn.
SCOTT, D. D...................535 W. Paul St., Rochester, N. Y.
SEAMAN, A. E..................................Houghton, Mich.

SEBENIUS, JOHN UNO.................Wolvin Bldg., Duluth, Minn.
SEEBER, R. R....................................Winona, Mich.
SEELYE, R. W..............................Sault Ste. Marie, Ont.
SELLS, MAX ....Florence, Wis.
SENTER, A. W....................................Hubbell, Mich.
SHELDEN, R. SKIFF..............................Houghton, Mich.
SHERLOCK, THOMAS ....Escanaba, Mich.
SHERRERD, JOHN M............340 Spring Garden St., Easton, Pa.
SHIELDS, IRVIN J..............................Houghton, Mich.
SIEBENTHAL, W. A..............................Republic, Mich.
SILL, GEO. A...................504 Marquette Bldg., Chicago, Ills.
SILVER, C. R.........................29 W. Lake St., Chicago, Ills.
SIMMONS, CHARLES ....Beacon, Mich.
SKINNER, MORTIMER B.558-560 W. Washington Blvd., Chicago, Ills.
SLAUGHTER, WILLIAM F..........................Vulcan, Mich.
SLINEY, DAVID J..............................Ishpeming, Mich.
SMITH, FRED ....Kearsarge, Mich.
SMITH, GEORGE M................................Duluth, Minn.
SMITH, WILLARD J..............................Mohawk, Mich.
SMITH, CARL G.................................Kearsarge, Mich.
SMITH, ALFRED L...............................Wakefield, Mich.
SMYTH, H. L........................Rotch Bldg., Cambridge, Mass.
SOADY, HARRY ....Duluth, Minn.
SORNBERGER, E. C.............208 Lancaster Ave., Buffalo, N. Y.
SPARKS, BENJAMIN F..............205 Ruby St., Houghton, Mich.
SPERR, F. W....................................Houghton. Mich.
SPORLEY, CHARLES L...........................Negaunee, Mich.
STAKEL, CHARLES J............................Ishpeming, Mich.
STANTON, F. McM....................208 5th Ave., New York City
STANTON, J. R.......................11 William St., New York City
STEPHENS, JAMES ....Ishpeming, Mich.
STEVENS, THOMAS J............................Ironwood, Mich.
STOEK, H. H...................University of Illinois, Urbana, Ills.
STRONG, CLARENCE G...........................Cincinnati, Ohio
SUESS, JOSEPH E...............................Negaunee, Mich.
SUTHERLAND, D. E..............................Ironwood, Mich.
SWAIN, RICHARD A.......General Electric Co., Minneapolis, Minn.
SWIFT, GEORGE D.................................Duluth, Minn.
SWIFT, PAUL D.................................Houghton, Mich.

TANCIG, A.......................................Hibbing, Minn.
TARR, S. W.......................610 Wolvin Bldg., Duluth, Minn.
TAYLOR, JAMES HALL....................Box 485, Chicago, Ills.
THIEMAN EDWARD ....Florence, Wis.
THOMAS, WILLIAM ....Ironwood, Mich.
THOMAS, KIRBY ....505 Pearl St., New York City
THOMS, REUBEN KNIGHT...............................Ely, Minn.

THOMPSON, G H....................................Hibbing, Minn.
THOMPSON, HENRY S................................Beacon, Mich.
THOMPSON, JAMES R.............................Ishpeming, Mich.
TILLINGHAST, EDWARD S.........................Hibbing, Minn.
TOWNSEND, C. V. R..............................Negaunee, Mich.
TRAVER, WILBER H....................Fisher Bldg., Chicago, Ills.
TREBILCOCK, JOHN ....Ishpeming, Mich.
TREBILCOCK, WILLIAM ....North Freedom, Wis.
TREPANIER, HENRY ....Iron Mountain, Mich.
TREZONA, CHARLES ....Ely, Minn.
TREVARROW, HENRY ....Negaunee, Mich.
TREVARTHAN, W. J..............................Bessemer, Mich.
TRIPP, CHESTER D........1515 Corn Exchange Bldg., Chicago, Ills.
TRUDGEON, JOHN ....Wakefield, Mich.
TUFTS, JOHN W.................900 Hackett Ave., Milwaukee, Wis.
TURNER, CHAS. N............Colby-Abbott Bldg., Milwaukee, Wis.
TYLER, CHARLES ....Negaunee, Mich.

UHLER, FRED WALTER...............................Buhl, Minn.
UREN, WILLIAM J.............124 College Ave., Houghton, Mich.

VALLAT, BENJAMIN W...........................Ironwood, Mich.
VAN DYKE, W. D................................Milwaukee, Wis.
VANDEVENTER, VIVIAN H.......................Ishpeming, Mich.
VAN EVERA, JOHN R............................Marquette, Mich.
VAN EVERA, WILBUR ....Virginia, Minn.
VAN MATER, J. A......................55 Wall St., New York City
VAN ORDEN, F. L...............................Houghton, Mich.
VILAS, P. M...........425 New York Life Bldg., Minneapolis, Minn.
VILAS, ROYAL L................................Ishpeming, Mich.
VIVIAN, JAMES G.................909 Alworth Bldg., Duluth, Minn.
VOGEL, F. A..........................25 Broad St., New York City

WADE, JEPTHA H.....................Wade Bldg., Cleveland, Ohio
WAGNER, JOHN M...............................Houghton, Mich.
WALKER, ROBERT L...............................Diorite, Mich.
WALKER, ELTON WILLARD ....Mass, Mich.
WALL, JAMES S...............................Iron River, Mich.
WALLACE, W. R.................................Houghton, Mich.
WALLACE GEORGE ....Marquette, Mich.
WARE, JOHN FRANKLIN.....Forest and Five Oaks Ave., Dayton, O.
WARE, FRED ....Negaunee, Mich.
WARREN, O. B...................................Hibbing, Minn.
WARRINER, S. D..............................Wilkesbarre, Pa.
WATSON, CHARLES H.........................Crystal Falls, Mich.
WEARNE, WILLIAM ....Hibbing, Minn.
WEBB, FRANCES J.............519 Providence Bldg., Duluth, Minn.

WELLS, PEARSON ....Ironwood, Mich.
WELKER, W. F......................................Ashland, Wis.
WENGLER, MATT P..........1055 Cambridge Ave., Milwaukee, Wis.
WESSINGER, W. E..................610 Wolvin Bldg., Duluth, Minn.
WEST, WILLIAM J..................................Hibbing, Minn.
WHEELWRIGHT, O. W..............................Florence, Wis.
WHITE, WILLIAM ....Virginia, Minn.
WHITE, EDWIN E..............................Ishpeming, Mich.
WHITEHEAD, R. G.................................Amasa, Mich.
WHITESIDE, DR. JOHN W.........................Ironwood, Mich.
WILLEY, NORMAN W........Shenango Furnace Co, Hibbing, Minn.
WILKINS, WILLIAM ....Detroit, Mich.
WILLIAMS, THOMAS H.................................Ely, Minn.
WILLIAMS, HERBERT H..........................Marquette, Mich.
WILLIAMS, PERCIVAL S............................Ramsay, Mich.
WILSON, EUGENE B...............................Scranton, Pa.
WINCHELL, HORACE V........505 Palace Bldg., Minneapolis, Minn.
WINTER, JOSEPH H..............................Negaunee, Mich.
WITHERBEE, F. S...........................Port Henry, New York
WOODWORTH, G. L...............................Iron River, Mich.
WOODWORTH, R. B..............427 Carnegie Bldg., Pittsburg, Pa.
WOOLF, PERCIVAL J..........................Minneapolis, Minn.
WORD, WILLIAM W..............60 Castro St., San Francisco, Cal.
WORDEN, EUCLID P.............................Milwaukee, Wis.

YATES, WILLIAM H.............................Negaunee, Mich.
YOUNG, H. OLIN ....Ishpeming, Mich.
YOUNGS, FRANK W.............................Iron River, Mich.
YOUNGS, G. W.................................Iron River, Mich.
YUNGBLUTH, A. J..............................Ishpeming, Mich.

ZAPFFE, CARL ......213 Citizens State Bank Bldg., Brainerd, Minn.

DEACEASED MEMBERS.

ARMSTRONG, J. F........1898
BAWDEN, JOHN T........1899
BENNETT, JAMES H.........
BIRKHEAD, LENNOX 1911
BROOKS, T. B............1902
BULLOCK, M. C............1899
COWLING, NICHOLAS ...1910
CONRO, ALBERT 1901
CLEAVES, WILL S.......1910
CHADBOURNE, T. L......1911
CUMMINGS, GEO. P......1911
DANIELS, JOHN 1898
DICKENSON, W. E.......1899
DOWNING, W. H..........1906
DUNCAN, JOHN 1904
DUNSTON, THOMAS B.......
GARBERSON, W. R.......1908
HALL, CHAS. H...........1910
HARPER, GEORGE V.....1905
HASELTON, H. S.........1911
HAYDEN, GEORGE 1902
HINTON, FRANCIS 1896
HOLLAND, JAMES 1900
HOLLEY, S. H...........1899
HOUGHTON, JACOB......1903
HYDE, WELCOME ....
JEFFERY, WALTER M....1906
JOCHIM, JOHN W........1905
KRUSE, JOHN C..........1907
LUSTFIELD, A............1904
LYON, JOHN B...........1900
MARR, GEORGE A........1905
MITCHELL, SAMUEL 1908
M'VICHIE, D..............1906
NINESE, EDMUND 1909
OLIVER, HENRY W......1904
PEARCE, H. A............1905
PERSONS, GEORGE R....1908
ROBERTS, E. S..............
ROWE, JAMES 1911
RYAN, EDWARD 1901
SHEPHARD, AMOS 1905
STANLAKE, JAMES 1910
STANTON, JOHN 1906
STURTEVANT, H. B......1910
THOMAS, HENRY 1905
TREVARTHEN, G. C......1898
TRUSCOTT, HENRY 1910
VAN DYKE, JOHN H.....1906
WALLACE, JOHN 1898
WHITE, PETER 1908
WHITNEY, J. D...........1894
WILLIAMS, W. H.........1897

LIST OF DECEASED MEMBERS REPORTED SINCE THE ANNUAL MEETING OF 1911.

MAAS, WM. J.................................December 4th, 1911
MILLER, A. M...............................February 20th, 1912
POPE, GRAHAM ....July 8th, 1912
STEVENS HORACE J.............................April 22nd, 1912
TOBIN, JAMES ....June 23rd, 1912

LIST OF PAPERS PUBLISHED IN PRECEDING VOLUMES.

1893—Vol. I.

1894—Vol. II.

1895—Vol. III.

1902—Vol. VIII.

1903—Vol. IX.

1904—Vol. X.

1905—Vol. XI.

1906—Vol. XII.

1908—Vol. XIII.

1909—XIV.

1910—Vol. XV.

1911—Vol. XVI.

LIST OF MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO AUGUST, 1912.

No.	Place.	Date.	Proceedings.
1	Iron Mountain, Mich	March 22-23, 1893	Vol. I
2	Houghton, Mich	March 7-9, 1894	Vol. II
3	Mesabi and Vermilion Ranges	March 6-8, 1895	Vol. III
4	Ishpeming, Mich	August 18-20, 1896	Vol. IV
5	Ironwood, Mich	August 16-18, 1898	Vol. V
6	Iron Mountain, Mich	February 6-8, 1900	Vol. VI
7	Houghton, Mich	March 5-9, 1901	Vol. VII
8	Mesabi and Vermilion Ranges	August 19-21, 1902	Vol. VIII
9	Ishpeming, Mich	August 18-20, 1903	Vol. IX
10	Ironwood, Mich	August 16-18, 1904	Vol. X
11	Iron Mountain, Mich	October 17-19, 1905	Vol. XI
12	Houghton, Mich	August 8-10, 1906	Vol. XII
13	Mesabi and Vermilion Ranges	June 24-27, 1908	Vol. XIII
14	Ishpeming, Mich	August 25-27, 1909	Vol. XIV
15	Ironwood, Mich	August 24-26, 1910	Vol. XV
16	Crystal Falls, Mich	August 22-24, 1911	Vol. XVI
17	Houghton, Mich	August 28-30, 1912	Vol. XVII

Note—No meetings were held in 1897, 1899 and 1907.

RULES OF THE INSTITUTE.

I.

OBJECTS.

The objects of the Lake Superior Mining Institute are to promote the arts and sciences connected with the economical production of the useful minerals and metals in the Lake Superior region, and the welfare of those employed in these industries, by means of meetings of social intercourse, by excursions, and by the reading and discussion of practical and professional papers, and to circulate, by means of publications among its members the information thus obtained.

II.

MEMBERSHIP.

Any person interested in the objects of the Institute is eligible for membership.

Honorary members not exceeding ten in number, may be admitted to all the privileges of regular members except to vote. They must be persons eminent in mining or sciences relating thereto.

III.

ELECTION OF MEMBERS.

Each person desirous of becoming a member shall be proposed by at least three members approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe), upon receiving three-fourths of the votes cast. Application must be accompanied by fee and dues as provided by Section V.

Each person proposed as an honorary member shall be recommended by at least ten members, approved by the Council, and elected by ballot at a regular meeting, (or by ballot at any time conducted through the mail, as the Council may prescribe), on receiving nine-tenths of the votes cast.

IV.

WITHDRAWAL FROM MEMBERSHIP.

Upon the recommendation of the Council, any member may be stricken from the list and denied the privilege of membership, by

the vote of three-fourths of the members present at any regular meeting, due notice having been mailed in writing by the Secretary to him.

V.

DUES.

The membership fee shall be five dollars and the annual dues five dollars, and applications for membership must be accompanied by a remittance of ten dollars; five dollars for such membership fee and five dollars for dues for the first year. Honorary members shall not be liable to dues. Any member not in arrears may become a life member by the payment of fifty dollars at one time, and shall not be liable thereafter to annual dues. Any member in arrears may, at the discretion of the Council, be deprived of the receipt of publications or be stricken from the list of members when in arrears six months; Provided, That he may be restored to membership by the Council on the payment of all arrears, or by re-election after an interval of three years.

VI.

OFFICERS.

There shall be a President, five Vice Presidents, five Managers, a Secretary and a Treasurer, and these Officers shall constitute the Council.

VII.

TERM OF OFFICE.

The President, Secretary and Treasurer shall be elected for one year, and the Vice Presidents and Managers for two years, except that at the first election two Vice Presidents and three Managers shall be elected for only one year. No President, Vice President, or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The term of office shall continue until the adjournment of the meeting at which their successors are elected.

Vacancies in the Council, whether by death, resignation, or the failure for one year to attend the Council meetings, or to perform the duties of the office, shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; Provided, That such appointment shall not render him ineligible at the next election.

VIII.

DUTIES OF OFFICERS.

All the affairs of the Institute shall be managed by the Council except the selection of the place of holding regular meetings.

The duties of all Officers shall be such as usually pertain to their offices, or may be delegated to them by the Council.

The Council may, in its discretion, require bonds to be given by the Treasurer, and may allow the Secretary such compensation for his services as they deem proper.

At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Five members of the Council shall constitute a quorum; but the Council may appoint an executive committee, business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary and recorded by him with the minutes.

There shall be a meeting of the Council at every regular meeting of the Institute and at such other times as they determine.

IX.

ELECTION OF OFFICERS.

Any five members not in arrears, may nominate and present to the Secretary over their signatures, at least thirty days before the annual meeting, the names of such candidates as they may select for offices falling under the rules. The Council, or a committee thereof duly authorized for the purpose, may also make similar nominations. The assent of the nominees shall have been secured in all cases.

No less than two weeks prior to the annual meeting, the Secretary shall mail to all members not in arrears a list of all nominations made and the number of officers to be voted for in the form of a letter ballot. Each member may vote either by striking from or adding to the names upon the list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing the ballot with his name, and either mailing it to the Secretary, or presenting it in person at the annual meeting.

In case nominations are not made thirty days prior to the date of the annual meeting for all the offices becoming vacant under the rules, nominations for such offices may be made at the said meeting by five members, not in arrears, and an election held by a written or printed ballot.

The ballots in either case shall be received and examined by three tellers appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected. The ballot shall be destroyed, and a list of the elected officers, certified by the tellers, shall be preserved by the Secretary.

X.

MEETINGS.

The annual meeting of the Institute shall be held at such time as

may be designated by the Council. The Institute may at a regular meeting select the place for holding the next regular meeting. If no place is selected by the Institute it shall be done by the Council.

Special meetings may be called whenever the Council may see fit; and the Secretary shall call a special meeting at the written request of twenty or more members. No other business shall be transacted at a special meeting than that for which it was called.

Notices of all meetings shall be mailed to all members at least thirty days in advance, with a statement of the business to be transacted, papers to be read, topics for discussion and excursions proposed.

No vote shall be taken at any meeting on any question not pertaining to the business of conducting the Institute.

Every question that shall properly come before any meeting of the Institute, shall be decided, unless otherwise provided for in these rules, by the votes of a majority of the members then present.

Any member may introduce a stranger to any regular meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

XI.

PAPERS AND PUBLICATIONS.

Any member may read a paper at any regular meeting of the Institute, provided the same shall have been submitted to and approved by the Council, or a committee duly authorized by it for that purpose prior to such meeting. All papers shall become the property of the Institute on their acceptance, and with the discussion thereon, shall subsequently be published for distribution. The number, form and distribution of all publications shall be under the control of the Council.

The Institute is not, as a body, responsible for the statements of facts or opinion advanced in papers or discussion at its meetings, and it is understood, that papers and discussions should not include personalities, or matters relating to politics, or purely to trade.

XII.

AMENDMENTS.

These rules may be amended by a two-thirds vote taken by letter ballot in the same manner as is provided for the election of officers by letter ballot; Provided, That written notice of the proposed amendment shall have been given at a previous meeting.

INFORMATION PREPARED FOR THE SEVENTEENTH ANNUAL MEETING OF THE

LAKE SUPERIOR MINING INSTITUTE

AUGUST 28TH, 29TH, AND 30TH, 1912

HELD IN THE COPPER COUNTRY WITH HEADQUARTERS AT HOUGHTON, MICHIGAN

COMPILED AND EDITED BY
ARTHUR L. CARNAHAN

Acknowledgment is extended to Dr. L. L. Hubbard for assistance in preparing the statements regarding geology, and to the mine executives and others for their co-operation.

THE HILLS OF HOUGHTON.

I stood on the hills of Houghton
As the sun sank slowly down,
With a farewell glance of gladness
O'er the grim old northern town,
And the south winds drifted softly
With a perfume strangely sweet,
And toyed with the native roses
That blossomed at my feet.

But I knew that the night was coming,
For the winds grew damp and chill
As they crept from the fragrant forest
And o'er the highest hill,
And a night bird sang in the valley,
A song that was tinged with woe,
A song they had sung in Eden,
That only the night birds know.

And out from the sister city
As deeper the shadows fell,
There rang o'er the murky waters
The tones of a silv'ry bell,
And the echoes, oh! the echoes,
Rolled gloriously and free,
As over the hills of Houghton
They clambered up to me.

Good night to the hills of Houghton,
The night bird's song has ceased
And a moon with a golden glory
Is gilding the distant east,
And the winds that bore the eagle
Afar on his fearless flight,
Lie hushed on the hills of Houghton,
Asleep in the sea of night.

—Allan Tyrrell.

COMMITTEES.

Local Committees in charge of arrangements for Copper Country meeting:

Finance Committee.

F. W. Denton, Chairman
C. L. Lawton
John Knox, Jr.
Willard J. Smith
R. M. Edwards
Dr. L. L. Hubbard

Transportation Committee.

Jas. B. Cooper, Chairman
James Robertson
F. R. Bolles
C. E. Webb
John C. Shields
Gardner Rogers
H. E. Stewart

Entertainment Committee.

James R. Dee, Chairman
W. D. Calverley
J. H. Rice
H. L. Baer
John L. Harris

Excursion Committee.

John Knox, Jr., Chairman
C. L. Lawton
C. H. Benedict
W. J. Richards
Willard J. Smith
S. Russell Smith

Reception Committee.

F. J. Bawden, Chairman
Henry L. Baer
Joseph R. Biscombe
Walter Bloomfield
Samuel Brady
Henry Brett
Frederick I. Cairns
William D. Calverley
Allen Cameron
M. J. Carroll
Benj. F. Chynoweth
James Chynoweth
Claude H. Cooper
James B. Cooper
James R. Dee
F. W. Denton
Robert P. Dunstan
A. D. Edwards
Herman W. Fesing
James Fisher, Jr.
Frank H. Getchell
George S. Goodale
H. S. Goodell
Norman W. Haire
John L. Harris
S. B. Harris
S. T. Harris
George L. Heath
J. H. Hodgson
A. B. Holtenhoff
R. E. Hore
L. L. Hubbard
Arthur A. Koch
George A. Koenig
David H. Ladd
C. L. Lawton
Thomas F. Lynch
Ronald H. MacDonald
James MacNaughton
Alfred H. Meuche
James A. MacDonald
F. W. McNair
F. W. Nichols
John Pentecost
W. G. Phillips
E. W. Presho
R. C. Pryor
J. T. Reeder
J. H. Reeder
W. J. Richards
F. W. Ridley
William V. Salvick
George P. Schubert
A. E. Seaman
A. W. Senter
Irving J. Shields
Fred Smith
Willard J. Smith
F. W. Sperr
H. L. Swift
Paul D. Swift
F. L. Van Orden
John M. Wagner
Edward S. Warne
W. J. Uren
A. H. Floeter
J. O. Peterson
David L. Vivian
Enoch Henderson
Edward S. Grierson
Edward Koepel
Allan F. Rees
James J. Byers
C. H. Moss
Robert H. Shields
William F. Miller
E. J. Dube
John T. MacNamara
John C. Pryor
James T. Healey
F. Corbin Douglass
Homer A. Guck
William B. Hoar
John C. Mann
R. B. Lang
R. Skiff Shelden
Jonah Orrison

ITINERARY.

Wednesday, August 28th.

Excursion to points south of Houghton:

Leave Houghton via Copper Range R. R.	2:00 p. m.
Arrive at Michigan Smelting Co., plant, Coles Creek	2:15 p. m.
Leave Coles Creek	3:00 p. m.
Arrive Painesdale	3:40 p. m.

NOTE—Those who prefer to do so, may stop at the Baltic mine to visit the shaft house, change house and oil house at that mine, while the remainder of the party continue on the trip to Painesdale to visit the compressor plant and the shops at that place. There will not be sufficient time to visit both places.

Leave Painesdale	5:00 p. m.
Arrive Houghton	6:00 p. m.

At 8 o'clock p. m., a business meeting will be held in the banquet hall of the new Masonic Building, Houghton.

Thursday, August 29th.

Excursion to points north of Houghton:

Leave Houghton, via Mineral Range R. R.	8:30 a. m.
Arrive Lake Linden	9:15 a. m.
Leave Lake Linden	10:30 a. m.
Arrive Ahmeek	11:30 a. m.
Leave Ahmeek	12:30 p. m.
Arrive Osceola	1:00 p. m.
Leave Osceola via Houghton County Street R'y	1:05 p. m.
Arrive Electric Park	1:15 p. m.
(Lunch will be served at Electric Park.)	
Leave Electric Park	2:35 p. m.
Arrive Boston (Franklin Jr., Mine)	2:50 p. m.
Leave Boston	3:50 p. m.
Arrive Houghton	4:30 p. m.

The members of the Institute and their guests are invited to attend a dinner. to be followed by a concert by the Calumet and Hecla band, at the Onigaming Yacht Club, Thursday evening at 6:30 o'clock.

A boat will leave the dock at the foot of Isle Royale street, Houghton, at 5 o'clock p. m., to convey the party to the club house.

Friday, August 30th.

8:30 a. m. to 10:30 a. m.—A business meeting of the Institute will be held in the Gymnasium building at the College of Mines.

10:30 a. m. to 1:00 p. m.—Demonstrations at the College of Mines.

Special excursions will be arranged for on request, if practicable. Application should be made to Mr. Robert H. Maurer, local secretary in charge of the Bureau of Information.

The Portage Lake Golf Club cordially extends the privileges of its club to the members of the Mining Institute and their guests. Arrangements may be made by applying to Mr. Robert H. Maurer, local secretary in charge of the Bureau of Information at his headquarters in the Douglass House, or to the Golf Club Committee, namely, Messrs. J. H Rice, W. D. Calverley, R. Skiff Shelden and John L. Harris.

GEOLOGICAL NOTES OF THE LAKE SUPERIOR COPPER FORMATION.

Measured from a line drawn between Keweenaw Point on on the south and Isle Royale on the north, the west end of the Lake Superior basin, southwesterly through Michigan, Wisconsin and Minnesota, is rimmed by a group of rock beds known geologically as the Keweenawan. Melaphyrs—old basic lava flows—and more acid rocks of similar and of intrusive origin also, were piled up one on another in successive layers at irregular time intervals during periods of volcanic activity.

Alternating irregularly with these melaphyrs are other beds, the accumulations of time intervals during which volcanic activity was dormant. In these intervals erosive agencies broke off fragments from the exposed rock surfaces, transported, rounded, ground up and gathered them into basins under water, forming them into conglomerates and sandstones.

Successive products of these erosive agencies, consolidated into beds, were in turn from time to time covered by other volcanic flows which became gradually less frequent and less voluminous, until they ceased altogether. And now on the inner rim of the basin on the lake shore we find gravel spread out between ledges of ancient rocks that jut into the water; and in quiet bays we find beaches of sand. These are the constituents of what may become another conglomerate horizon in the group, if volcanic agencies shall ever again spread over it their mantle of desolation.

Just as a knowledge of the stars helps us to realize the immensity of space, so does the thickness of the Great Conglomerate bed that runs down Keweenaw Point, and is best

exposed near the west shore between Eagle River and Copper Harbor, help us to form an idea of the ages required to complete some of the geological changes that have taken place in this area first visited by white men only 250 years ago. This conglomerate is in places 2,200 feet thick. What eons must it have taken to round off and accumulate this bed, which is only one in the Keweenawan series whose total is from 22,000 to 48,000 feet thick!

That a much larger area was once covered by Keweenawan beds is evident from the patches of similar rock left on the north or Canadian shore, on islands, and on the extreme east shore of the lake.

The Keweenawan series lies above the Huronian, which is exposed in the Fall River near L'Anse; and below the Potsdam which is well shown in the quarries of Portage Entry on the east side of Keweenaw Point. It is chiefly remarkable as the seat of the largest known deposits of native copper in the world. The beds of the series everywhere dip almost universally towards Lake Superior at different angles, being generally flatter near the lake shore.

The lava beds—commonly called traps—at several horizons carry in their upper amygdaloidal and scoriaceous or fractured portions, metallic copper from the minutest particles up to masses of several hundred tons weight. Only one conglomerate bed has thus far been found to carry copper in commercial quantity, the famous Calumet conglomerate. In the early days of mining from 1845 through a period of years, veins crossing the lava and conglomerate beds at different angles and containing large masses of copper, were the only deposits wrought here. But these veins were shallow and were rapidly worked out, while improved methods of mining soon created a more lasting industry in the beds themselves, with their less massive and more widely disseminated contents, which are now being profitably mined to a depth of over a mile.

The beds of the Keweenawan series of Michigan run near-

ly parallel with the course of the lake shore. From erosion of the softer strata, from slide-faulting or shearing, or from both, several prominent ridges rising to heights of 800 or 900 feet are developed in the harder rocks. Of these the Greenstone bluff in Keweenaw County, and the Evergreen bluff in Ontonagon County are the most interesting features of the landscape. Erosion across the harder beds on the lake shore and laterally behind these beds in the softer formations, have formed several picturesque and roomy harbors, which offer safe anchorage to the smaller water craft and attractive resorts for summer guests.

HISTORICAL NOTES.

Copper was taken from the Lake Superior region for the uses of man far back in the ages of unrecorded history. Proof of this is in the wide distribution of copper and "half-breed" silver-copper throughout uncivilized America, which evidently came from these shores; and the ancient mine pits, stone hammers and other relics of the prehistoric miners. Explorers of 70 years ago and later found copper in hiding places, where it had been left by the native workers, and the ancient pits frequently guided white men in their search for copper. The pits, stone hammers, copper knives and fish hooks and occasional charcoal remnants, probably the remains of crude calcining processes, are still in evidence.

Knowledge of these copper deposits came to white men soon after Columbus' discovery. Included in notable references to this are descriptions by Giovanni Verazzani in 1524 of copper beadstones worn by natives along the New England coast and plates of wrought copper in their possession; and the narratives of Captain Jacques Cartier, who found copper in the hands of the savages along the St. Lawrence River in 1535.

The natives were evasive when pressed to tell whence the copper came, although those more remote from its source may not have known. Samuel de Champlain, writing of his ex-

pedition which set out from Quebec June 14, 1610, says he met "two savages. They came on board my barque, where I gave them good entertainment. * * * The Algonquin savage * * * drew from a fack a piece of copper a foot long, which he gave me. * * * He gave me to underftand that there were large quantities where he had taken this which was on the bank of a river, (the Ontonagon?) near a great lake. He said they gathered it in lumps." On his map of 1632 is shown a lake evidently intended for Superior, although woefully undersized, showing an island "Where there is a mine of copper."

The source of the native copper remained a secret from the white race for nearly 170 years after Columbus. In September, 1660. Father Rene Mesnard was brought by friendly Indians from Sault de Ste. Marie to Keweenaw Bay and established a mission near the present village of L'Anse, at the very neck of Keweenaw Peninsula. Copper is not mentioned in the letters to his superiors. The next summer he started for the northwest and was lost and died in the forest August 7 or 8

The first white men to set foot on Keweenaw Peninsula, so far as we know today, were the brothers-in-law Peter Esprit Radisson and Medard Chouart des Grosilliers. In the summer of 1661, at the time Mesnard was so close to the peninsula and met his tragic end, they were skirting along the south shore of Lake Superior. Radisson in writing from memory several years later describes a section that conforms in part to the mouth of the Ontonagon or Iron Rivers, but he places it between Sault de Ste. Marie and the Pictured Rocks, a locality that yields no copper. It may be that his memory tricked him, or that this was a hiding place for copper or a deposit of float copper.

He says, "We found a small river. * * * They named me it pauabickkomefibs, w^{ch} fignifieth a fmall river of copper. I asked him the reason. He told me, "Come and I fhall fhow thee y^{e} reason why." I was in a place w^{ch} was

not 200 paces in y^e wood, where many peeces of copper weare uncovered. Further he told me that the mountaine I ſaw was of nothing elſe." The name given to Radisson for the river resembles in its first part the word pewabic, which is the Indian word for iron and not copper, and suggests that this stream may have been our modern Iron River, west of the Ontonagon, along whose banks copper would be likely to lie.

The narrative then describes the Pictured Rocks, where Radisson named the "Portall of S^t. Peter because my name is so-called," and the Huron Islands, which he named "The Trinity, there being three," after which his "thwarting" of Keweenaw Peninsula is narrated. "We came to the mouth of a ſmall river, where we killed ſome Oriniacks (moosse). We found meddows * * * as smooth as a board. We * * * found ſome pools made by Castors (beavers) * * * w^{ch} drowned more than 20 leagues * * * of soyle. * * * Herein grows moſſe w^{ch} is 2 foot thick or there abouts, and * * * if you take not great care ſink downe to yo^r head. * * * I ſpeake by experience for I meſelfe have bin catched often. * * * I must w^{th} my hands hold the moſſe and go ſo like a frogg. * * * Having passed that place, we made a carriage through the land for two leagues. The way was well beaten becauſe of the comers and goers, who by making that paſſage ſhortens their paſſage by 8 days by touring about the point that goes very farr in that great lake. * * * In the end of that point, that goeth very farre, there is an iſle, as I was told, all of copper. This I have not ſeene."

Thenceforth mention of copper became more frequent among the writers and the location more specificly given. Captain Jonathan Carter predicted that copper would be transported down the great lakes, and Alexander Mackenzie gave a roseate opinion of the copper resources. These and other reports during the 100 years after Radisson aroused the adventurers. The first attempt to commercialize the district, of which we have record, was by Alexander Henry. During

years of successful Indian trading and stirring adventures his experiences had carried him along the chain of the Great Lakes and in 1765 he saw the famous mass of copper in the Ontonagon River. In 1771 he returned with a crew and equipment furnished by English capital, and left the men to work through the winter. They drove an adit 40 feet into the frozen clay bank and found frequent masses of copper, but with the spring thaw their "mine" was in trouble. The entire project was abandoned and Henry in writnig his biography 40 years later said that "The copper ores of Lake Superior can never be profitably sought for except for local consumption. The country must be cultivated, and peopled, before they can deserve notice. The copper mines of Lake Superior have been more than once represented to the world in colors capable of deceiving fresh adventurers, and the statement in the text will not have been uselessly made, if it should at any time serve as a beacon to the unwary."

His judgment was strikingly correct. The country had to be cultivated and peopled before the copper deposits deserved notice He could not foresee the railroad and steam navigation, which alone have made it possible to work these deposits at a profit. Without these advantages profitable copper mining in the region would be impossible. This has widened the zone of Henry's idea of "local consumption" and the "country to be cultivated and peopled."

Public appreciation of the copper resources began when Governor Cass of Michigan personally conducted an expedition through the region in 1820. Henry R. Schoolcraft accompanied him as mineralogist. The next aggressive step was in 1832, when Schoolcraft conducted a similar expedition, bringing with him Dr. Douglass Houghton as geologist. Schoolcraft on his first visit observed that the float copper came from rock matrixes and Houghton found copper bedded in the rock. Apparently white men had not discovered this before but the ancients knew it and extracted the copper from its rock matrix.

Houghton expressed the opinion that the "trap rock formation was the source of the masses of copper." He visited the region three times during 1832 and 1833 and his interest both in the scientific and commercial sides of its resources was unflagging. Five years later he was appointed the first State Geologist and his reports brought a large influx of prospectors in 1843 and later years. After a career of great usefulness, during which he presented to the world the true conditions of Keweenaw Peninsula, he was drowned off the shore of Eagle River in 1845. Probably the first explorers to arrive were Jim Paull and Nick Minicler, who came from Wisconsin in 1843, enduring the hardships of winters, reaching here in March. The government opened a land office that year at Copper Harbor and the commercial exploitation of the region was fairly started. There was great activity and writers describe animated scenes. Viewed from the lake, as the voyager traveled in his canoe, the landscape was dotted with tents in many places from Keweenaw Point to the Porcupine Mountains. We owe a graphic picture of those times to Rev. Pietzel, a Methodist missionary. Hardships, privation, and very nearly starvation were among the experiences of the pioneers at times when new financing was needed or severe winters delayed the arrival of the boats in the spring.

To recount the names of the leaders of this pioneer work and tell of the successive discoveries of the ore bodies, lies more within the functions of the Biographical Committee of this Institute than the limited scope of a brief guide book. The practical experiences of the early mining companies are exhaustively described in the reports of Foster and Whitney, published by the United States government in 1850 and 1851.

The growth from those days of uncertainty and indifferent support to the present period of colossal production, overawing depths and gigantic machinery, is a history too familiar to repeat yet too pregnant in events to be adequately recorded.

SMELTING PROCESSES.

No. 0 (the richest) mineral, No. 1 mineral, mass copper

and barrel copper, and in some smelter practices No. 2 mineral, are charged into the reverberatory furnaces for melting and refining. Occasionally they are melted in one furnace and refined in another. The product of these furnaces is (1) skimmings or slags and (2) merchantable copper. The latter is cast into marketable shapes. The slags are charged into a cupola (blast) furnace. The product of this furnace is (1) waste slag or tailings and (2) black copper (96 per cent pure; known as cupola blocks). The waste slag is discharged into the lake. The cupola blocks are charged into reverberatory furnaces for melting and refining. The product of these furnaces is (1) skimmings or slags and (2) anodes and merchantable copper. The anodes are in suitable shapes for electrolytic treatment and shipped mainly to the Buffalo smelter.

No. 2 (low-grade) mineral is charged into reverberatory furnaces for melting. The product of these furnaces consists of (1) skimmings or slags and (2) buttons (accumulations of metallic copper under the skimmings of reverberatory furnaces). The slags go to the cupola furnace and are treated as the slags described above, yielding anodes and merchantable copper. The buttons are treated the same as cupola blocks. At the Calumet & Hecla the greater portion of the conglomerate No. 1 mineral (the richest product from the mill jig) is shipped to the Buffalo smelter.

In electrolytic refining at the Buffalo smelter and elsewhere, anodes are dissolved by electrolysis, which yields two products —(1) slimes, which fall to the bottom in the electrolytic bath, and (2) cathodes. The object of this treatment is two-fold, (1) to remove arsenic and other deleterious elements and (2) to extract silver. The arsenic remains in solution in the bath and the silver falls with the slime, to be separated by fire refining. At Buffalo No. 1 conglomerate mineral and the cathodes are charged into the reverberatory furnaces for melting and refining, the process thenceforth being the same as described above.

COMPOUND HOISTING ENGINE.

An interesting work of reconstruction is in progress on Quincy's No. 8 hoisting engine. This engine at present is simple-steam, direct-acting composite drum (cylindrical center and tapered ends) with a winding capacity of 5,000 feet, and duplex steam cylinders, each 32 inches in diameter and 72-inch stroke. The cylinders are to be removed and duplex tandem cylinders installed, the steam to be used compound and condensing. Diameter of high pressure cylinders 20 inches; low pressure 38 inches; stroke 72 inches. The old drum and crank shaft will be replaced with a new and larger crank shaft and a composite drum having a maximum diameter of 20 feet and a winding capacity of 5,600 feet. The condenser will be operated by a 12x36 inch horizontal Corliss simple engine A new boiler plant will be installed to furnish steam at 150 pounds. The old boiler plant will be removed to No. 9 shaft and the old drum and cylinders will be fitted to new frames at No. 9 shaft.

EQUIPMENT.

The following pages contain descriptions of all the large equipments and many of the smaller plants of the district. It would be impracticable to list all items, but space is left in the sections devoted to the respective mines, wherein additional notes may be entered.

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C
Hoisting Engines:				
Rock Hoist	No. 11 Hecla	Lake Shore	Duplex simple	N-C
Rock Hoist	No. 10 Hecla	Iron Bay	Duplex simple	N-C
Pewabic	No. 9 } 1 bdg.	Leavitt-Morris	(a) Triple expan.	C
Hancock	No. 7	Identical with Pewabic		
Man Hoist	No. 7 } 1 bdg.	Leavitt-Morris	(b) Compound	C
Man Hoist	No. 8	Identical with No. 7 man hoist		
Man Hoists Nos. 9, 6 & 2 Hecla, Nos. 2 & 6 Calumet each has man hoist identical with				
Seneca	No. 8 } 1 bdg.	Leavitt-Morris	(c) Triple expan.	C
Houghton	No. 6	Identical with Seneca. They also serve		
Marquette-Chippewa	No. 4 Cal.	Corliss	(d) Compound	C
Minnesota-Escanaba	No. 5 Cal.	Rice & Sargent	(d) Compound	C
Illinois-Wisconsin	No. 6 Cal.	Identical with Minnesota-Escanaba		
Minong-Siscowit	Red Jacket	Leavitt-Morris	(e) Triple-expan.	C
Mesnard-Pontiac	Red Jacket	Identical with Minong-Siscowit. Both in one		
Delaware-Iroquois	Red Jacket	Corliss	(d) Compound	C
Rock Hoist	R. J. Slope	Gen. Elec. Nordberg	Motor driven; operates at	
Rock Hoist (2)	Nos. 13 & 16	Ident. with Ahmeek	Central No. 1 hoist. engine	
Rock Hoist	No. 14 Amg.	Lidgerwood	Duplex simple	N-C
Detroit-Onota	No. 15 Amg.	Corliss	(d) Compound	N-C
Rock Hoist	No. 17 Amg.	Identical with No. 14 hoisting engine.		
Rock Hoist	No. 18 Amg.	Identical with No. 11 hoisting engine.		
Rock Hoist	No. 21 Kear.	Lidgerwood	Duplex simple	N-C
Stamp Heads (11)	Cal. mill	Morris	Simple cush'n'd	C
(1)	Hecla mill	Nordberg	Steeple comp'nd	C
(16)	Identical with 11 heads in Calumet mill.			
Air Compressors:				
Mackinac		Lev-Mor.-Nord.	(z) Triple expan.	C
Superior	Superior	Leavitt-Morris	Rocker comp'nd	C
Baraga	Eng. house	Leavitt-Morris	Simple	C
Rockland		Corliss	Simple	C
Frontenac	Frontenac	Leavitt-Morris	Rocker comp'nd	C
La Salle	Eng. house	Corliss	Simple	C
Perrott		Corliss	Simple	C
Pumping Engines:				
Electric	L.S.W.W.	Mor.-Gen. Elec.	Motor driven	Cent.
Pepin	L.S.W.W.	Snow (tandem)	4cy., 3ple.-expan.	C
Nippissing	L.S.W.W.	Worthington	Duplex comp'nd	
Minnehaha	Pond W.W.	West Point	Rocker comp'nd	C
Nipigong	Pond W.W.	Worthington	Tan. dup. comp.	C
Portage	Pond W.W.	Worthington	Tan. dup. comp.	C
Heriot	Pond W.W.	Snow	3 crank 3ple ex.	C
Michigan	Mill	Leavitt-Morris	Vertical 3ple ex.	C
Huron	Mill	Brown-Morris	Simple	C
Arcadian	Mill	Brown-Morris	Tandem comp'nd	C
Ontario	Mill	Leavitt-Morris	Vertical comp'nd	C
Wabeek	Mill	Leavitt-Morris	Vertical 3ple ex.	C
Power Engines:				
Ontonagon	Mill	Leavitt-Morris	3 cyl. comp'nd	C
Osage	Mill	Identical with Ontonagon.		
Owego	Mill	Identical with Ontonagon.		
Saginaw	Mill	Allis-Chalmers	Dup. vert. comp.	C
Gratiot	Mill	Leavitt-Morris	Vertical 3ple ex.	C
New Turbine	Mill	Allis-Chalmers	Mixed flow turbine	

CALUMET & HECLA

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Stroke	Style	Action (o)	Brake	Diam.	Grooved for	Win'g Capa'y (f)	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
14			18	Cyldr.	Gr. 7	(g)	5		775	5000	5000	1
20			36	Cyldr.	Gr. 16	(g)	12		4000	6000	5000	1¼
20⅛	31¾	50	48	(k)	(n)	(h)	25		5000	12000	15000	1⅜
14		(2)24	30		Gr.23¾	(g)	16		4824	6000	30men	1⅜
7; separately housed												
18	27¾	48	90	(k)	Gr. 25	(h)	25		5000	12000	15000	1⅝
Hecla and No. 2 Calumet												
(2)17		(2)34	60	(k)	Gr. 25	(h)	25		5000	(p)	20000	1⅝
(2)18		(2)36	60	(k)	Gr. 25	(h)	25		5000	12000	20000	1⅝
20½	31¾	50	72	(2 cylinders of each dimension)					Whiting system. (x)			
Each operates 2 skips in balance												
(2)16		(2)32	48	Drum identical with Minn.-Escanaba.								
57th level underground												
18			24	Cyldr.	Grd.	(g)			1000	8000	10000	1¼
(2)18		(2)32	48	Drum identical with Minn.-Escanaba. Detroit-Onota under erection								
14			18	Drum identical with No. 11 hoisting engine.								
21½		14	24	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
15½		32	24									
29	51¼	(2)58	90	(y)		2	34½		54	60	9180	
40		70	72	(aa)								
40			60	(aa)								
30			48	(aa)								
27¾		48	72									
30			72									
30			48									
650 h.p., 1300 v.,				750				3,000,000*; 800′ head				
17	29½	(2)34	36	40	4 7¾x36 plungers,			1,500,000*; Idle				
(2)19¼		(2)33⅜	24	30	4 8¾x24 plungers,			1,000,000*; Idle				
11¾		24	48	32	1 17 x48 plunger,			3,000,000*; 100′ head				
(2)21		(2)42	36	25	2 19½x36 plungers,			10,000,000*; 100′ head				
(2)14		(2)24	36	25	2 20 x36 plungers,			2,000,000*; 100′ head				
18	33	54	36	50	6 19 x36 plungers,			15,000,000*; 100′ head				
18	27¾	48	90	30	2 34 x48 plungers,			60,000,000*; 90′ head				
18			42	18	x36 plungers,			20,000,000*; 72′ head				
13		26	42	18	x36 plungers,			20,000,000*; 72′ head				
18		24	60	34	18 x33 plungers,			20,000,000*; 60′ head				
18	27¾	48	60	58				70,000,000*;				
24		(2)40	48	Drives generator, 2000 k.w.				These furnish current for 240 motors ranging up to 700 h.p.				
				Drives generator, 2000 k.w.								
				Drives generator, 2000 k.w.								
(2)17		(2)40	48	Drives generator, 1000 k.w.								
18	27¾	48	90	Drives generator, 1000 k.w.								
(Under contract but not built) Will furnish maximum of 10,000 k.w.												

Boilers Plants:		
Superior	(14)	4 Morris; 4 Dickson; 2 Kendall & Roberts; 2 Atlantic; 2 Lake
Frontenac	(6)	Return tube type, each 750 h.p.
Houghton-Seneca	(5)	3 Lake Erie; 2 Atlantic; of locomotive twin furnace type, each
Hancock-Pewabic	(5)	Of locomotive twin furnace type, each 750 h. p.
Red Jacket	(4)	Of locomotive twin furnace type, each 750 h. p.
Pond W. W.	(3)	Of cylindrical return tube type.
Lake Sup. W. W.	(4)	Of cylindrical return tube type.
Lake Linden	(20)	Babcock & Wilcox water tube type, each 500 h.p.

Sand Wheels (3)	Diameter, 51′ 4¼″; width, 11′; 244 buckets, 4′x2′ 4″; shaft 10′ per second at inner edge of bucket, 30,000,000 gals. water
Sand Wheel (1)	Diameter, 62′ 6¾″; width, 11′ 9″; 544 buckets, 4′ 3½″x2′9″; of bucket; 40,000,000 gals. water and 8,000 tons sand in 24

(a) 3-crank rocker. (b) 3-crank rocker; triple cylinder. (c) Rocker. (d) Duplex tandem.
(e) Rocker; duplex. (f) per layer; may overlap indefinitely. (g) Hand; band.
(h) Hydraulic; band. (k) Cylinder; clutch. (n) Mortise geared, 30″ diameter.
(o) Gear diameter in inches.

ADVENTURE

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines (2)	Nos. 1 & 3	Allis	Duplex simple	N-C	
	No. 5	Development equipment.			
Stamp Heads (3)	Mill	Fraser-Chalmers	Simple		
Air Compressors	No. 3	Rand	Cross comp'nd	C	150
	New No. 5	Rand	Simple	N-C	150
Pumping Engine	Mill	Fraser-Chalmers	Cross comp'nd	C	150

Boiler Plants	No. 1 3 locomotive type No. 3 5 locomotive type	No. 5 2 locomotive type Mill 5 locomotive type

Crushing Plants

No. 1 steel rockhouse, 2 18x24″ crushers, 2 13x20″ crushers
No. 3 steel rockhouse, 2 18x24″ crushers, 2 13x20″ crushers

CALUMET & HECLA

Erie; of locomotive type, each of 750 h.p.

750 h.p.

23′ 6″x22″; elevates water and sand 43′ 2″; estimated weight, 400,000 lbs.; capacity at and 4000 tons sand in 24 hrs.; pinion is belt driven from motor.

shaft, 26′ 5¼″x25″; elevates water and sand 54′; capacity at 10′ per second at inner edge hrs.; pinion is rope driven from motor.

(p) 2, 6,000 lb. timber trucks in tandem; estimated timber weight 20,000 lbs.
(x) Rope carried 3 times around 2 tandem 19′ sheaves and through 312′ tail race.
(y) 3 2-stage compressors driven by one triple expan., quadruple cylinder engine.
*Gals. per 24 hrs. (z) Rocker; 4 cylinder.
(aa) 1 compressor identical with Mackinac; and other smaller compressors driven by Superior engine; Baraga and Rockland are auxiliaries.

ADVENTURE

STEAM CYLINDER

High Pr	Inter Pr	Low Pr	Stroke	Style	Action	Brake	Diam.	[illegible]	[illegible]	[illegible]	[illegible]	Rope Diam.
24			36	(b)	Direct	(a)				5000	12000	
20			25									

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Stroke	Free Air per Min.
24		44	48	77	[illegible]	2	42		28½	48	[illegible]
16			24	94	[illegible]		[illegible]			24	[illegible]

(a) Steam; shoe. (b) 2 cones.

ALLOUEZ

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines	No. 1 shaft	Identical with Ahmeek Central No. 1 hoist. engi			
	No. 2 shaft	Allis	Duplex simple	N-C	100
Stamp Heads (4)	L.M.S.&R.Co.	Nordberg	Steeple compound	C	150
(2)	"	"	Simple	C	150
Air Compressor	No. 1 shaft	Laidlow-Dunn-Gor'n	Cross compound	C	145
Pumping Engine	L.M.S.&R.Co.	Nordberg	Triple Expans'n	C	150
Power Engine	L.M.S.&R.Co.	Nordberg	Cross compound	C	150
	" "	Taylor-Chandler	Simple	N-C	150
Boiler Plants	No. 1	6 locomotive type, 215 h.p., 84", 144 tubes.			
	No. 2	2 locomotive type, 165 h.p., 72", 106 tubes.			
	L.M.S.&R.Co.	4 locomotive type, 215 h.p., 84", 144 tubes.			
Crusher Plants	No. 1 steel rockhouse; 3 jaw type crushers, 18"x24".				
	No. 1 steel rockhouse; 1 jaw type crusher, 24"x48".				

CENTENNIAL

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	No. 1 shaft	Indentical with Ahmeek No. 1 Hoisting Engine			
	No. 2 shaft	Sullivan	Duplex simple	N-C	100
Air Compressors	No. 2	Laidlow-Dunn-Gor'n	Cross comp'nd	C	120
	No. 2	Rand	Cross comp'nd	C	100
Pumping Engine	L.M.S.&R.Co.	owned jointly with Allouez, which see.			

Boiler Plants 6 locomotive type; 215 h.p.; 84"; 144 tubes.
L.M.S.&R.Co. owned jointly with Allouez, which see.

Crusher Plants No. 1 frame rockhouse; 3 18"x24" crushers.
" " No. 2 steel rockhouse; cylindrical bin; 2 18"x24" crushers.

ALLOUEZ

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Stroke	Style	Action	Brake	Diam.	Grooved for	Win'g Capa'y	Skip (Lbs.)	Rope (Lbs.)	Rope Diam.
(2)30			72	(a)	Direct	(b)			4000	6000	10000	1¼

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str	Free Air per Min. (cu. ft.)
15½		32	24								
20			24								
25		54	60	46		2	32		50	60	6000
13½	25	38	44	40	3 plungers 18½×44; 15,000,000*; 85' head						
16		36	36	160	Indicated horse power 500						
11			12	275	Indicated horse power 120	Drives generator 75 k w					

(a) Tapered ends. (b) Steam; shoe *Gallons per 24 hrs

CENTENNIAL

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Stroke	Style	Action	Brake	Diam.	Grooved for	Win'g Capa'y	Skip (Lbs.)	Rope (Lbs.)	Rope Diam.
34			60	Cylic	Direct	[illegible]				[illegible]	[illegible]	1¼

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str	Free Air per Min. (cu. ft.)
27		56	60	[illegible]	80 80	2	[illegible]		[illegible]	60	[illegible] [illegible]

AHMEEK

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines	Central No. 1	Nordberg	Duplex simple	N-C	115
	Central No. 2	Same as Ahmeek Central No. 1 hoisting engine			
	North No. 3	Same as Ahmeek Central No. 1 hoisting engine			
	North No. 4	Same as Ahmeek Central No. 1 hoisting engine			
Air Compressors	Central	Nordberg	Cross compound	C	150
	Central	Nordberg	Cross compound	C	150
	North	Nordberg	Triple Expan.	C	150
Pumping Engine	Mill	Nordberg	Triple Expan.	C	150
Power Engines	Central	Nordberg	Tand. Comp'nd	C	150
	Mill	Nordberg	Cross comp'nd	N-C	150
Stamp Heads (4)	Mill	Nordberg	Steeple comp'nd	C	150

Boiler Plants
- Central — 8 Locomotive type, 200 h.p.; steel stack 7' 6"x150'.
- Mill — 6 Locomotive type, 200 h.p.; steel stack.

Crusher Plant — No. 1 and 2 shafts tram rock to central plant; after crushing, it is elevated by link belt system to cylindrical bin for loading.

AHMEEK

STEAM CYLINDER

High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
32			72	(b)	Direct	(a)	18½		5031	5300	6000	1⅜
						AIR CYLINDER						
				Rev. per Min.	Air Pr.	Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
20		36	44	82		2	21		35	44	4000	
20		40	44	82		2	21		35	44	4000	
22½	41	60	48	75		2	(3)19		(3)30	48	8830	
22	40	60	52	35		3 plungers 29x52; *40,000,000; 90′ head.						
13		26	36	120		Drives generator 200 k.w.						
18		30	36	100		Drives generator 300 k.w.						
16½		32	24	108								

*Gallons per 24 hrs. (a) Steam; shoe. (b) Tapered ends.

COPPER RANGE CONSOLIDATED

EQUIPMENT		LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines	(2)	Baltic 2&4	Nordberg	Duplex Direct	N-C	100
		Baltic No. 3	Fraser-Chalmers	Duplex simple	N-C	100
		Baltic No. 5	Allis-Chalmers	Duplex simple	N-C	100
	(2)	Cham. D&E	Same as Baltic No. 2 hoisting engine.			
	(2)	Cham. B&C	Same as Ahmeek Central No. 1 hoisting engine.			
	(3)	Trim. 2,3,4	Same as Ahmeek Central No. 1 hoisting engine.			
		Atlantic A	Allis-Chalmers	Duplex simple	N-C	100
		Atlantic B	Fraser-Chalmers	Duplex simple	N-C	100
		Atlantic D	Same as Ahmeek Central No. 1 hoisting engine.			
Man Hoist		Atlantic D	Bullock	Duplex simple	N-C	100
Air Compressors		Baltic No. 2	Nordberg	Cross compound	N-C	150
		Baltic No. 3	Riedler-Fr.-Ch.	Cross-compound	N-C	150
		Champ. B	Ingersoll-Rand	Simple (tand. air)	N-C	150
		Champ. F	Nordberg	Quadruple Expan.	C	300
		Trim'n No. 2	Rand	Cross-compound	C	
		Trim'n No. 3	Nordberg	Triple Expan.	C	200
Pumping Engines		Champ. mill	Nordberg	Triple-Expan.	C	200
		Trim'n mill	Nordberg	Triple Expan.	C	175
Power Engines		Baltic mill	Nordberg	Cross-compound	C	150
		Champ. mill	Identical with Baltic mill power engine			
		Trim'n mill	Nordberg	Cross compound	N-C	170
Stamp Heads	(2)	Baltic mill	Nordberg	Steeple-comp'd	C	150
	(4)	Baltic mill	Nordberg	Simple	C	150
	(2)	Champ. mill	Identical with Baltic compound heads			
	(4)	Champ. mill	Identical with Baltic simple heads			
	(4)	Trim'n mill	Identical with Baltic compound heads			

Boiler Plants		Type	h.p.			Type	h.
No. 2 Baltic	3	Stirling	(j) 250	No. 3 Trim'n	4	Stirling	2
No. 3 Baltic	4	Stirling	250	No. 4 Trim'n	2	Locomotive	1
No. 5 Baltic	1	Locomotive	100	B Champion	4	Locomotive	1
No. 5 Baltic	1	Locomotive	75	C Champion	2	Locomotive	1
No. 2 Trim'n	4	Locomotive	125	D Champion	2	Locomotive	1

Crushing Plants	Baltic Mine 3 steel rockhouses, square bins; 1 frame rockhou Champion Mine 4 steel rockhouses, s, square bins. Trimountain Mine 3 steel rockhousesquare bins.
Smelting Plant	Part ownership in Michigan Smelting Co.

(a) Double cone. (b) Cylinder. (c) Steam; shoe. (d) Hand; band. (e) Steam; band. (f) Tapered ends. *Gallons in 24 hours.

COPPER RANGE CONSOLIDATED

STEAM CYLINDER High Pr	Inter Pr	Low Pr	Stroke	DRUM Style	Action	Brake	Diam.	Grooved for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
24			60	(f)	Direct	(c)	14⅛		3000	5600	9000	1⅛
24			60	(a)	Direct	(c)			4400	8200	10000	1¼
14			18	(b)	Geared	(d)	6					
26			48	(a)	Direct	(e)		Idle				
24			60	(b)	Direct	(e)	12	Idle				
								Idle				
24			60	(b)	Direct		12	Idle				

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)
19		34	44	70		2	25		38	44	4032
22		38	48	70		2	24		38	48	4000
18			24	100		2	24¼		13	24	1115
14½	22-38	54	48	75		2	(2)23		(2)37	48	8960
18		34	36	70		2	20		30½	36	2400
14	26	43	42	80		3	21	28½	38	42	4400
14	28	43	44	41		3 plungers 21⅛x44; *20,000,000; 68′ head					
17	35	48	44	41		3 plungers 21⅛x44; *20,000,000; 72′ head					
14		32	30	125							
14		24	36	90							
15½		32	24	105							
20			24	105							

		Type	h.p.
E Champion	2	Locomotive	150
F Champion	3	Geary	250
F Champion	1	Parker	300
Baltic mill	6	Stirling	(g) 250
Champ. mill	4	Stirling	(h) 200
Champ. mill	3	Marine	(h) 150
Champ. mill	2	Marine	(h) 200
Trim'n mill	6	Stirling	(k) 250

(j) Equipped for superheating.

(g) Concrete stack 7′ 10″x210′.

(h) Induced draft.

(i) Steel stack 7′x180′.

(k) Steel stack 7′x180′.

FRANKLIN

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engine	Main shaft	Nordberg Edwards system; see paper on subject.	Duplex direct	N-C	
Stamp Heads (2)	Mill	Cuyahoga	Simple	C	125
(3)	Mill	Allis	Simple	C	125
Air Compressor	Mine	Ingersoll-Rand	Cross comp'nd	C	165
Pumping Engines	Mill	Allis	Vertical comp.	C	125
	Underg'd	Nordberg	Cross comp'nd	C	150
Power Engines	Mill	Allis	Simple	N-C	125
	Mill	Ide-West Elec.	Simple	N-C	125

Boiler Plants
Mine, 3 210 h.p., 1 80 h.p., locomotive type.
Mill, 4 Stirling.

Crusher Plant
Steel rockhouse, 1 cylindrical bin, 1 24x36" steel crusher.

HANCOCK

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	No. 2	Sullivan	Duplex simple	N-C	150
	No. 2	Sullivan	Duplex simple	N-C	150
Air Compressors	No. 2	Sullivan	Tand. comp'nd	C	145
	No. 2	Ingersoll-Rand	Cross-comp'nd	C	145
	No. 2	Sullivan	Direct	N-C	145
Generator Set	No. 2	General Electric	Motor driven; 50 k.w. 200		
Boiler Plants	No. 2	8 Kewaunee type, 150 h.p.	Concrete stack 2'9" and		
Crusher Plant	No. 2	Steel rockhouse, 2 cylindrical bins, 2 24x36' crushers			

(a) Clutch; 2 cylinder drums.

FRANKLIN

STEAM CYLINDER							DRUM					
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
46			72	(a)	Direct		15		5000	4000	20000	1⅝
20 20			24 24	Rev. per Min.	Air Pr.	Stages	AIR CYLINDER High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
					75	2					3500	
24 8		48 16	40 20	27 75		2 37½x40 plunger, 15,000,000*; 55' head. 2 $2\frac{5}{16}$--3¼x20 plungers, 150,000*; 1680' head						
16 8½			42 10	57 300		Drives Generator, 25 k.w., 210 am., 114 v.						

*Gals. per 24 hrs. (a) Cylinder.

HANCOCK

STEAM CYLINDER							DRUM					
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
36 24			72 48	Cyldr. (a)	Direct Direct	Band Band	15 8		4900 4000	10000 4000	16000 6000	1½ 1¼
				Rev. per Min.	Air Pr.	Stages	AIR CYLINDER High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
18		34	30	100	80	2	18		30	30	2450	
16		28	20	150	80	2	15		25	20	1700	
22			24	110	80	1	14½			24	1380	
250 v.				1200		Alternating current converted to direct current, to drive 2 tramming motors.						
7'6"x126												

ISLE ROYALE

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	No. 2	Same as Ahmeek Central No. 1 hoisting engine			
	No. 4	Same as Ahmeek Central No. 1 hoisting engine			
	No. 5	Allis	Duplex simple		
	No. 6	Same as Ahmeek Central No. 1 hoisting engine			
Stamp Heads (3)	Mill	Nordberg	Steeple comp.	C	
Air Compressors	No. 2	Nordberg	Cross-comp'nd	C	150
	No. 4	Nordberg	Cross comp'nd	C	100
	No. 6	Same as Ahmeek Central compressor.			
Pumping Engine	Mill	Nordberg	Triple Expan.	C	150
Boiler Plants	No. 2	No. 4	No. 5	No. 6	

Crushing Plants

No. 2 Steel rockhouse, square bins. 2 18x24 crushers.
No. 4 Steel rockhouse, cylindrical bin. 1 24x36 crusher.
No. 5 Frame rockhouse. 1 18x20 crusher.
No. 6 Steel rockhouse, cylindrical bin. 1 24x36 crusher.

*Gals. per 24 hrs.

LAKE

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engine	New shaft	Identical with Ahmeek Central No. 1 hoisting			
Air Compressor	Main plant	Nordberg	Cross-comp'nd	C	150

Crushing Plant Steel rockhouse.

ISLE ROYALE

STEAM CYLINDER						DRUM						
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
20			60						3500	5000	10000	1¼
15½		32	24				AIR CYLINDER				Free Air per Min. (cu. ft.)	
				Rev. per Min.	Air Pr.	Stages	High	Inter	Low	Str.		
19		37	44	68		2	21½		35	44	3340	
18		32	42	80		2	17¾		27	42	2200	
15	28	42	44			3 plungers, 18¾x44", 16,000,000*; 100' head						

Smelting plant; part ownership with Tamarack and Osceola.

LAKE

STEAM CYLINDER						DRUM						
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
engine. (16,000 lbs. rock).												
							AIR CYLINDER				Free Air per Min. (cu. ft.)	
				Rev. per Min.	Air Pr.	Stages	High	Inter	Low	Str.		
22		42	48	73	80	2	26		40	48	5000	

OSCEOLA

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines (5)	Osceola No. 5 and 6 shafts, North Kearsarge No. 1,3 & 4 shafts				
(2)	S.K.Nos.1&2	Allis			
Air Compressors	N.K.No. 3	Nordberg	Cross comp'nd	C	150
	N.K. No. 4	Identical with Ahmeek Central air compressor.			
	N.K. No. 1	Allis-Nordberg	Cross comp'nd	C	150
	S. Kears.	Allis-Nordberg	Cross comp'nd	C	110
	S. Kears.	Allis-Nordberg	Cross comp'nd	C	110
	Osc. No. 6	Identical with N. Kearsarge No. 3 compressor.			
Pumping Engines	Mill	Nordberg	Triple expan.	C	150
	Mill	Allis	Vertical comp'nd	C	150
Power Engines	Osc. No. 6	Nordberg	Tandem comp.	C	150
	N. K. No. 3	Identical with Osceola No. 6 power engine.			
	Mill	Fraser-Chalmers	Tandem comp.	C	150
Stamp Heads (7)	Mill	Nordberg	Steeple comp.	C	150
Boiler Plants	Mill, 11 locomotive type, 250 h.p. steel stack 14x155'. North Kearsarge, 13 locomotive type. South Kearsarge, 5 locomotive type. Osceola, 6 locomotive type.				
Crusher Plants	Osceola, 2 frame rockhouses. South Kearsarge, 1 frame rockhouse. North Kearsarge No. 1, steel rockhouse in construction. North Kearsarge No. 3, frame rockhouse. North Kearsarge No. 4, steel rockhouse; cylindrical bin.				

*Gals. per 24 hrs.

OSCEOLA

STEAM CYLINDER							DRUM					
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
same as Ahmeek Central No. 1 hoisting engine.												

High Pr	Inter Pr	Low Pr	Str-oke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)
22		42	48	69		2	26		40	48	4800
		38	48	75		2			38	48	3340
		30	42	75		2			18	42	
		36	42	75		2			36	42	1180
22	40	60	52	35	3 plungers, 29x52", 40,000,000*; 80' head.						
			38	26	3 plungers, 36x38", 15,000,000*; 80' head. (Idle						
9		18	36	120	Drives generator 150 k.w.						
16		20	34	83	Drives generator 100 k.w.						
15½		32	24	102							

Smelting plant; part ownership with Tamarack and Isle Royale.

MASS

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	B C	Fraser-Chalmers Fraser-Chalmers	Duplex simple Duplex simple	N-C N-C	125 125
Stamp Heads (2)	Mill	Nordberg	Simple	C	
Air Compressor	B	Rand	Cross comp'nd	C	125
Pumping Engine	Mill	Nordberg	Triple Expan.	C	150
Boiler Plants	B C Mill	2 Stirling 250 h.p. 2 Locomotive type, 150 h.p. 2 Stirling 250 h.p.			
Crushing Plants	B Frame rock house, 18x24' crusher. C Steel rock house, 18x24' crusher.				

(a) Steam; band. (b) 2 cylinder. (c) Single cone. (d) 10½-8½. *Gals. per 24 hrs

MICHIGAN

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	A B C	Web.-Camp.-Lane Allis-Chalmers Web.-Camp.-Lane	Duplex-simple Duplex-simple Duplex-simple	N-C N-C N-C	110 110 110
Stamp Head	Mill	Nordberg	Steeple comp'nd		
Air Compressor	A-B	Ingersoll	Cross-comp'nd	C	110
Power Engine	Mill	Wisconsin	Cross comp'nd	C	
Boilers	Mine Mill	4 Locomotive type, 165 h.p. each. 3 Water tube, 203 h.p. each.			

(a) Tapered ends. (b) Steam; shoe. (c) 2 Cylinder.

MASS

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs,)	Rope Diam.
				(b)	Direct	(a)	10½		3000		5000	1¼
				(c)	Direct	(a)	(d)		2000		9000	1¼
20			24									
				45	85	2						
13½	25	38	44	41		3 plungers, 18¾x44; 16,000,000*; 80′ hea						

MICHIGAN

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
24			60	(a)	Direct	(b)	16-7		3500	3000	6000	1⅛
24			60	(c)	Direct	(b)	8		3500	3000	6000	1¼
10			13	Cyldr.	Direct	(b)	6					1
				Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
36		34¼	19	2		22½		42	19		3500	
15		30	36									

MOHAWK

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engine	No.2 pl't(1)	Nordberg	Duplex simple	N-C	125
	No.2 pl't(2)	Identical with No. 2 plant (1) hoisting engine			
	No. 3	Fraser-Chalmers	Duplex simple	N-C	125
	No. 4	Nordberg	Duplex simple	N-C	125
	No. 5	Bullock	Duplex simple	N-C	125
	No. 6	Identical with No. 2 plant (1) hoisting engine			
Stamp Heads (4)	Mill	Nordberg	Steeple comp'nd	C	
Air Compressor	No. 4	Nordberg	Cross comp'nd	C	150
	No. 4	Ingersoll-Sargent	Cross comp'nd	C	150
Pumping Engine (c)	Mill				
(c)	Mill				
Power Engine	Mill	Nordberg	Cross comp'nd	C	140

Boiler Plants No. 2 3 locomotive type, 200 h.p. No. 4 5 locomotive type, 200 h.p.
No. 5 2 locomotive type, 200 h.p. No. 6 2 locomotive type, 200 h.p.
Mill 5 Stirling type, 200 h.p. (c)

Crusher Plants Nos. 1, 2, 3, and 5, frame rockhouses; No. 4 steel rockhouse; each 1 18x24 and 2 smaller crushers.
No. 6 plans steel rockhouse, 1 24x36 crusher.

OJIBWAY

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines (2)	No. 1 & 2	Nordberg	Duplex simple	N-C	
Air Compressor	Central	Ingersoll-Rand	Simple	C	
	Central	Ingersoll-Rand	Motor driven		
Power Engine	Central	Russell	Simple	C	

Boiler Plant 2 locomotive type, 150 h. p.

MOHAWK

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Stroke	Style	Action	Brake	Diam.	Grooved for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
24			60	(a)		(b)	18½		4000	4000	10000	1¼
				Cyldr.			12		1500	4000	4000	1⅛
24			60	(a)		(b)	14⅛		3200	4000	10000	1⅛
				Cyldr.					2700	6600	10000	1⅛
(Not yet delivered)												

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)
15½		32	24								
22		42	48	72		2	26		40	48	5000
22		46	48	70		2	27¼		42¼	48	5260
10		20	36	130		Drives generator 150 k.w.					

Smelting plant; part ownership in Mich. Smelting Co.

*Gals. per 24 hrs. (a) Tapered ends. (b) Steam; shoe.
(c) Owned jointly with Wolverine, which see.

OJIBWAY

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Stroke	Style	Action	Brake	Diam.	Grooved for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
–				Cyldr.	Geared		5'				4000	⅞

High Pr	Inter Pr	Low Pr	Stroke	Rev. per Min.	Air Pr.	AIR CYLINDER Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)
						2					1200
						2					750
				275	Drives generator 100 k.w. which furnishes power for compressor, pumps, shops and light.						

QUINCY

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines	Nos. 7, 2, 6, 8, and 9 shafts.				
Air Compressors (6)	In four plants; combined capacity, 27,000 cu. ft. of free ai				
Pumping Engines (2)	Mill				
Stamp Heads (7)	Mill				
Boiler Plants	Mill 14 boilers; mine 32 boilers.				
Crusher Plants	No. 7 steel rockhouse, square bin. No. 2 frame rockhouse.				

QUINCY

'EAM CYLINDER							DRUM						
gh Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
				Rev. per Min.	Air Pr.	Stages	AIR CYLINDER High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	

r minute; fitted for 100 lb. air pressure.

. 6 steel rockhouse, cylindrical bin.No. 8 frame and steel rockhouse.

SUPERIOR

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engine	No. 1	Allis	Duplex simple	N-C	
Compressors (2)	No. 1	Sullivan	Simple		
Boiler Plant 3 locomotive type, 750 h.p.					

(a) Hand; band.

VICTORIA

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engine	Main shaft	Web.-Camp.-Lane	Duplex simple		
Stamp Head (1)	Mill	Cuyahoga	Simple		
Air Compressor	Hydraulic system; water descending vertical shaft entrains air				
Boiler Plants	Battery for heating and emergency. All machinery driven by				
Crushing Plant 1 frame rockhouse.					

SUPERIOR

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
24			60			(a)			4000			1¼

						AIR CYLINDER				
				Rev. per Min.	Air Pr.	Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)
22			24				14½			24	

VICTORIA

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
				1-cone	Direct							

						AIR CYLINDER				
24			26	Rev. per Min.	Air Pr.	Stages	High	Inter	Low	Str.	Free Air per Min. (cu. ft.)

which is trapped and compressed by the weight of the water.

compressed air.

WINONA

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C	Boiler Press.
Hoisting Engines	No. 3 No. 4 No. 1 K. P.	Steam driven Electric driven			
Stamp Heads (2)	Mill	Fraser-Chalmers	Simple		
Air Compressors	No. 3 Slpg. Riv.	Nordberg Nordberg	Cross comp'nd Triple expan.	C C	100 200
Pumping Engine					
Power Engine	No. 3 generating station, which furnish current for lighting a				
Boiler Plants	No. 3				
Crusher Plants	No. 3 and 4 steel rockhouses.				

WOLVERINE

EQUIPMENT	LOCATION	BUILDER	TYPE	Cond. Non-C.	Boiler Press.
Hoisting Engines	No. 3 No. 4	Allis Nordberg	Duplex simple Duplex simple	N-C N-C	125 125
Air Compressors	No. 4 No. 4	Rand Rand	Cross comp'nd Cross comp'nd	C C	125 125
Pumping Engines (d) (d)	Mill Mill	Snow Nordberg	Triple expan. Triple expan.	C C	175 175
Boiler Plants	No. 3 3 firebox type, 150 h.p. No. 4 3 Stirling type, 150 h.p. Mill 4 Stirling type, 200 h.p. (2 owned jointly with Mohawk				
Crusher Plants	No. 3, steel rockhouse, 1 18x24, 2 smaller crushers. No. 4, frame rockhouse, 1 18x24, 2 smaller crushers.				

(a) Tapered ends. (b) Steam; band. (c) Steam; shoe. *Gals. per 24 hrs.

WINONA

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
				Rev. per Min.	Air Pr.	Stages	AIR CYLINDER High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
18		32	42	82	80	2	18		29	42	2900	
15½	32	48	44	80	80	3	21	29	40	44	5000	

for hoisting engines.

WOLVERINE

STEAM CYLINDER				DRUM								
High Pr	Inter Pr	Low Pr	Str-oke	Style	Action	Brake	Diam.	Grov-ed for	Win'g Capa'y	Skip (Lbs.)	Rock (Lbs.)	Rope Diam.
24			60	Cyldr.	Direct	(b)	17		4000	6600	10000	1¼
24			60	(a)	Direct	(c)	18½		4500	5000	8000	1¼
				Rev. per Min.	Air Pr.	Stages	AIR CYLINDER High	Inter	Low	Str.	Free Air per Min. (cu. ft.)	
18		34	36	72	65	2	20		30	36	2000	
14		28	36	72	65	2	16		26	36	1800	
18	33	54	36	42		3, 22 x36' plungers, 20,000,000*; 90' head.						
11	22	32	36			3, 13⅝x36' plungers, 8,000,000*; 90' head.						

Smelting plant; part ownership in Mich. Smelting Co.

(d) Owned jointly with Mohawk.

TIME TABLES

D. S. S. & A.

Through Trains Leave Houghton as Follows:

For Negaunee, Ishpeming, Marquette, Sault Ste. Marie, and all points east via Mackinaw, 11.00 A. M.

For Negaunee, Ishpeming, Marquette, Escanaba, Green Bay, Milwaukee, Chicago, and all points via C. & N. W. Ry., 3:00 P. M.

For Champion, Crystal Falls, Iron Mountain, Green Bay, Milwaukee, Chicago, and all points via C. M. & St. P. Ry., 4:45 P. M.

For Marquette, Duluth, Sault Ste. Marie, and all points via Mackinaw, 11:00 P. M.

Local Trains Leave Houghton as Follows:

For Hancock and Calumet, 9:25 A. M., 12:05 P. M., and 7:55 P. M. with connections at Hancock for Dollar Bay and Lake Linden.

COPPER RANGE

Through Trains Leave Houghton as Follows:

For Mass, Rockland and Ontonagon 9:20 A. M.

For Mass, Rockland, Ontonagon, Iron Mountain, Green Bay, Milwaukee, Chicago and all points via C. M. & St. P. Ry., 9:35 P. M.

Local Trains Leave Houghton as Follows:

For Dollar Bay, Lake Linden and Calumet, 7:00 A. M., 3:50 P. M., 7:00 P. M.

For Atlantic, South Range, Trimountain and Painesdale, 7:10 A. M., 9:30 A. M., 12:50 P. M., 2:25 P. M., 5:30 P. M., 11:35 P. M.

For Atlantic, Redridge, Beacon Hill and Freda, 8:30 A. M., 12:10 P. M.

HOUGHTON COUNTY TRACTION CO.

Out of Houghton:

Cars for Calumet leave East Houghton 5 and 35 minutes after the hour and pass the Douglass House 7 minutes later. Connections for Mohawk are made on cars leaving 35 minutes after the hour and for Wolverine or Lake Linden on cars 5 minutes after the hour.

Cars for West Hancock pass the Douglass House four times an hour.

Out of Calumet:

Cars for Houghton leave on the even hour and half hour.

Cars for Mohawk leave 10 minutes before the hour.

Cars for Wolverine leave 10 minutes after the hour.

Cars for Lake Linden leave 15 minutes before the hour.

THE SEVENTEENTH ANNUAL MEETING.

WEDNESDAY, AUGUST 28TH, 1912.

The Seventeenth Annual Meeting of the Institute was called in the Copper Country with headquarters at Houghton, Mich., upwards of two hundred and fifty members and guests being in attendance. The program arranged by the local committees provided trips to various parts of the district, covering as far as possible the later developments. Leaving Houghton by special train via the Copper Range railroad at 2 o'clock, the first stop was at the Michigan smelting works where the visitors arrived at the "pouring" hour and the process of drawing the molten copper from the furnaces and casting it into merchantable shapes was witnessed. At the Champion mine where the next stop was made, interest centered in the "F" power plant, inspecting the quadruple expansion air compressor with its regenerative feed-water heating system, with a capacity of 9,000 cubic feet of free air per minute, a plant which holds the world's record for steam economy. The boiler plant adjoining is equipped for 250-pound steam pressure. "F" location also has a vertical cross compound electric generator of 250 kilowatt capacity.

At the Baltic mine the dry and change house and its sanitary arrangements were the features of interest. The oil house, of modern construction for convenience and economy in receiving and distributing oils, was carefully inspected. Leaving the Baltic the party returned to Houghton at 6 o'clock p. m.

EVENING SESSION.

The evening session was held in the banquet hall of the new Masonic Building. The meeting was opened by President

F. W. Denton, who extended a very cordial welcome to the members. Mr. Denton spoke in part as follows:

PRESIDENTIAL ADDRESS.

This is the seventeenth annual meeting of this Institute, and it is the hope of the local members that those participating in it will have some of the pleasure and profit that has attended its predecessors. The program and the itinerary, are, I think, fully set forth in the printed matter that has been furnished. This particular time finds the mining interests in the Copper Country in a hopeful mood. We are hopeful that the present profitable price of our product will continue long enough for us to cash in a good profit on the year's business, with perhaps a little to spare for developments. Throughout the iron region affairs are quite active and while prices may not be so satisfactory, still that region, too, is prosperous.

Since the last meeting of the Institute here, six years ago, perhaps the most notable development has been along the eastern border of the range. In that region deposits of copper have been opened which have added very materially to the value of that side of the range and have enlarged very much the prospects of the district.

From the operating standpoint the change that calls for first mention, perhaps, is in the milling department, where the improvement in fine grinding machinery has made it possible to extract a little more of the copper from the rock. The application of fine grinding is now gradually being extended, and next year a great deal more will be done than is now being done. Along with that has come the development of cheaper power. At present, two companies have plans under way for the installation of low pressure turbines. By means of them an effort will be made to extract more power from the fuel consumed at the stamp mills by using the exhaust steam from the stamp heads. If hopes in that district are realized, we shall have a horse-power at considerably less cost than can now be obtained, and the grinding process may be still further extended.

Underground a great deal of attention is being paid here, as in the iron regions, to improvements in machine drills, and while results are still indefinite and it is hard to say just what the outcome will be, I think it safe to predict a permanent improvement.

In other directions there is steady progress, particularly in matters pertaining to the welfare of employes. As in the iron region, most of the copper companies will adopt the provisions of the new employes' accident compensation law. The effects of this may be far reaching, and in connection with it, a still more strenuous effort is being made to check accidents.

PROGRAM.

Papers were presented in the order as they appear in the program. Mr. Crowell being unable to attend the meeting, his paper was read by title:

*Method of Sampling at Lake Superior Mines, by Benedict Crowell, Cleveland, Ohio.

The next paper entitled *System of Safety Inspection of The Cleveland-Cliffs Iron Co., by William Conibear, was presented in oral abstract by the author. The discussion is published following the paper.

The following papers were read by title:

*Raising Shaft at Rolling Mill Mine, Negaunee, Mich., by Edwin N. Cory, Negaunee, Mich.

*Mine Sanitation, by E. B. Wilson, Scranton, Pa.

Raising, Sinking and Concreting No. 3 Shaft Negaunee Mine, by S. R. Elliott, Negaunee, Mich.

The following papers were presented in oral abstract by the authors, and the discussion is published with the papers:

The Unexplored Parts of the Copper Range of Keweenaw Point, by Alfred C. Lane, Tufts College, Mass.

*Footwall Shafts in the Lake Superior Copper Mines, by Dr. L. L. Hubbard, Houghton, Mich.

This concluded the reading of papers for this session.

*Distributed in printed form.

The next order of business was the appointing of special committees. On motions duly made and carried the President appointed the following committees, to report at the regular session on Friday morning:

COMMITTEE ON NOMINATIONS—M. M. Duncan, Ishpeming, Mich.; William Kelly, Vulcan, Mich.; Luther C. Brewer, Ironwood, Mich.; James B. Cooper, Hubbell, Mich.; William J. West, Hibbing, Minn.

AUDITING COMMITTEE—John M. Wagner, Houghton, Mich.; G. S. Barber, Bessemer, Mich.; Wm. P. Chinn, McKinley, Minn.

The President announced that the meetnig of the Council would be held Thursday afternoon at 5 o'clock at the Houghton Club.

Mr. Kelly then presented the following resolution:

MR. KELLY: At the last meeting of the Institute a year ago I presented a resolution which was carried at the meeting, and which reads as follows:

> "Resolved that the Council be authorized to appoint, from time to time, special committees to consider and report to the Institute through the council upon such subjects as changes in mining, safety devices, the securing of data or papers on mining methods, definition of mining terms, affiliation with other societies and such other subjects as the council shall deem it desirable to inquire into, such reports not to be binding on the Institute except action is taken by the Institute in accordance with the rules."

At the meeting of the Council in Ishpeming about a month ago this resolution was considered. The Council thought it exceedingly desirable to appoint at this time some committees on some of these subjects. It was suggested also that some of these committees would entail expenses that should probably be defrayed by the Institute, I therefore desire to make a motion that the Chair appoint a committee of three to draft an amendment to the rules which will put the substance of this resolution in such form that it will be in accordance with the organic law of the Institute, that the

Council may not only appoint such committees but that they should also have authority to expend money for such purposes. The Council, as I understand it, feel that without such definite authority they would not be authorized to make any expenditures.

I think I may add that some of these subjects are exceedingly desirable to have considered by the Institute. The question of safety in the mines is very properly a subject that should have definite consideration. By this means the Institute may be able to obtain information that will add to our knowledge and help in solving the questions involved.

Motion carried.

The President appointed the following committee: William Kelly, Vulcan, Mich.; W. H. Johnston, Ishpeming, Mich., and F. W. McNair, Houghton, Mich.

MR. HEARDING: I move that a committee of three be appointed by the Chair to adopt suitable resolutions on the death of Graham Pope who was President of this Institute in 1901.

Motion carried.

MR. DENTON: I am sure we are all in sympathy with this motion; we certainly counted, a few months ago, on having Mr. Pope with us today, and he has always taken a great deal of interest in the Institute and has done a great deal of work for its welfare.

The President appointed the following committee: J. H. Hearding, Duluth, Minn.; James B. Cooper, Hubbell, Mich.; M. M. Duncan, Ishpeming, Mich.

An adjournment was then taken until Friday morning, at 8:30 o'clock at the Gymnasium building of the Michigan College of Mines.

THURSDAY, AUGUST 29TH, 1912.

The party left Houghton by special train at 8:30 a. m., via Mineral Range railroad for Lake Linden, arriving at the Calumet & Hecla stamp mills, where the first stop was made. The visitors were under the guidance of the mill officials.

In the electrical power house, which is in the group of plants at the stamp mills, the five generating units with a combined capacity of 8,000 kilowatts invited admiration. In the extension to this building, for which the foundation is just being finished, the new mixed-flow turbine generator, one unit of 10,000 kilowatt capacity, will be installed. The current from the present plant drives 240 motors, ranging in capacity up to 700 horsepower. The automatic stoking boiler plant, the milling operations and the regrinding plant received close attention.

At Ahmeek the visitors found several carloads of mass copper on the tracks, ready for shipment to the smelter. On one car were two masses which constituted a carload, each weighing probably 15 tons. The Ahmeek method of surface tramming from No. 1 and No. 2 shafts to a central crushing plant and elevating by link-belt conveyor to a cylindrical bin, whence the crushed rock is loaded to cars for shipment to the mill, was one of the features of interest. Another was the set of concrete "timbering" adjusted on surface for inspection exactly as it is placed in the mine. This method of shaft lining is used throughout the Ahmeek mine, which is the first company to develop it to the complete exclusion of timber. Description of this form is given in the excerpt by E. B. Jones in the paper on "Some Applications of Concrete Underground," by H. T. Mercer, published in this volume.

Leaving the Ahmeek the party was taken to Electric Park, for an hour's recreation, and refreshments. Franklin, Jr., was then visited, the feature of interest being the new air-balanced hoisting engine for operating the single skip with its 10-ton load of rock. The operation of this plant is fully described in the paper by R. H. Corbett on the "Description of an Air-Balanced Hoisting Engine, Franklin Mining Company," published in this volume. The system was invented by R. M. Edwards, general manager of Franklin, and this is the first plant of the kind ever built.

After the return to Houghton the members with their

guests were taken by boat to the Onigaming Yacht Club where dinner was served. The Calumet & Hecla band accompanied the party and the musical feature added much to the splendid entertainment afforded.

Following the coffee and cigars President Denton introduced Dr. Holmes, director of the Bureau of Mines, of the Department of the Interior, who spoke on the two leading functions of the Bureau, the safe-guarding of the lives of miners and the reduction of the wastes of the mining industry. This is the first year in which an appropriation has been made to apply the work of the Bureau to metal mining, previous attention during the short time since it was established having been confined largely to coal mining. The appropriation for metal mining, although only $50,000, serves as the foundation for a work that promises vast possibilities.

Dr. Holmes' visit was in response to an invitation of President Denton and was accepted with the object of bringing the activities of the Lake Superior Mining Institute into cooperation and support of the Bureau's work. Dr. Holmes spoke in part as follows:

Following the lines of technical discussion of the Institute, the Bureau of Mines wants to gather all the information possible regarding conservation of life and of resources in mining and give it out through channels where it will do the greatest good. Company secrets are not wanted and the Bureau refuses absolutely to have relations with the courts. Its officials have been called upon to testify in cases of mine accidents where Bureau investigations have been made directly after the catastrophy, but they have steadfastly refused to comply and have been supported in this by the attorney general and the courts themselves.

As a result of the Bureau's work there has been a distinct diminishing of disasters. Besides seeking to induce the managements to introduce every possible safety precaution the Bureau stimulates discipline, self-respect and moral development among the men with the object of developing greater caution,

and practical instruction is given in the handling of explosives and other departments of mining which become dangerous if carelessly performed. Mine rescue work is taught.

Regarding the conservation of mineral resources, the iron mining companies are now laying aside and otherwise preserving ores that at present are too lean for commercial value, against the time when these grades of ores will be needed and used. Exhaustive research has shown that the western hemisphere has very little real coal resources either north or south of the United States, and in time this country must supply fuel to those countries whose future development will demand this commodity.

If the United States can export coal to South America in exchange for the products of that continent which we need, great good will be accomplished. To this purpose it behooves our coal operators to save the 250,000,000 tons of coal now wasted yearly by bad mining, a half ton being wasted for every ton extracted. The yearly waste of natural gas, one of our most important and readily available fuels, is greater than the total yearly consumption of manufactured gas in all the plants of the country. Conservation of mineral resources does not mean that we shall stint ourselves. The needs of the country are increasing faster than the population. Nevertheless we may use all we need. But it is a sacred obligation to posterity that we preserve and leave what we do not use. The vastness of the mineral industries was made impressive by the statements that crude minerals constitute 60 per cent of the freight of the country, that $2,000,000,000 worth of ores and minerals are manufactured into $20,000,000,000 worth of commodities and 2,000,000 men are employed in the various departments.

A vote of thanks was extended to Dr. Holmes for his very interesting address. It is regretted that it could not be published in full.

FRIDAY, AUGUST 30TH, 1912.

The members assembled in the Gymnasium building of the College of Mines where the meeting was called to order

at 9:00 a. m., President Denton, presiding. The presentation of papers was continued. Mr. Wilson, whose paper on Mine Sanitation was read by title on Wednesday evening, presented abstracts of the same with some additional matter pertaining to the subject.

The next paper, by Edward Koepel, Beacon Hill, Mich., on the subject of *Construction of Intakes at the Mills of the Champion and Trimountain Companies, was read by title, Mr. Koepel being absent.

Balancing Rock Crushers, by O. P. Hood, Pittsburg, Pa., was read, the paper not having been printed in advance.

*Some Applications of Concrete Underground, by H. T. Mercer, Painesdale, Mich., was read by title and the discussion is published following the paper.

The following papers were presented in oral abstract by the authors and the discussion is published with the papers:

*In the Lake Superior Area—What Influence If Any, Did the Thickness and Contour of the Footwall Beds Have Upon the Subsequent Deposition and Distribution of Copper in Overlying Beds, by Dr. L. L. Hubbard, Houghton, Mich.

Failures of the Rules of Following the Hanging in the Development of Lake Superior Copper Mines, by F. W. Sperr, Houghton, Mich.

The following papers were read by title:

*Rock House Practice of the Quincy Mining Company, by T. C. DeSollar, Hancock, Mich.

*Rock House Practice of the Copper Range Consolidated Company, by H. T. Mercer, Painesdale, Mich.

*The New Franklin Hoist, by R. H. Corbett, Hancock, Mich.

*Economical Lubrication, by W. W. Davis, Boston, Mass.

Notes of Methods of Mining Iron Ore in the Lake Superior District, by F. W. Sperr, Houghton, Mich.

This concluded the reading of papers and the meeting proceeded with the routine business, reports of Council and Committees.

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*Distributed in printed form.

REPORT OF THE COUNCIL.

Secretary's report of Receipts and Disbursements from August 27th, 1911, to August 26th, 1912.

RECEIPTS.

Cash on hand August 24th, 1911			$5,771 17
Entrance fees for 1911	$ 135 00		
Dues for 1911	1,885 00		
Back dues	140 00		
Advance dues for 1912	35 00		
Sale of Proceedings	47 75		
Sale of Institute badges	8 00		
Total		$2,250 75	
Interest on deposit		184 25	
Total receipts			2,435 00
Grand total			$8,206 17

DISBURSEMENTS.

Stationery and printing	$ 40 00		
Postage	133 09		
Freight and express	19 44		
Telephone and telegraphing	6 38		
Secretary's salary	750 00		
Stenographic work	60 00		
Total		$1,008 91	
Publishing Proceedings XVI	831 50		
Photographs, maps, etc	147 18		
Advance papers, 1911	142 75		
Badges for meeting, 1912	81 25		
Total		1,202 68	
Total disbursements			2,211 59
Cash on hand August 22nd, 1912			5,994 58
Grand total			$8,206 17

MEMBERSHIP.

	1912.	1911.	1910.
Members in good standing	437	467	465
Honorary members	4	4	4
Life members	2	2	2
Members in arrears (2 years, '09-'10)	43	44	36
Total	486	517	507
New members admitted	31	46	45
New members not qualified	4	3	1
New members added	27	43	44

TREASURER'S REPORT.

Treasurer's Report from August 22nd, 1911, to August 26th, 1912:

Cash on hand August 22nd, 1911	$5,771 17	
Received from secretary	2,250 75	
Received interest on deposits	184 25	
Paid drafts issued by secretary		$2,211 59
Cash on hand August 26th, 1912		5,994 58
Totals	$8,206 17	$8,206 17

COUNCIL.

A meeting of the Council of the Lake Superior Mining Institute was held at Ishpeming, on Tuesday, July 23rd, at 2:30 p. m., and the following members were present: F. W. Denton, E. W. Hopkins, L. C. Brewer, J. B. Cooper, J. E. Jopling, A. J. Yungbluth.

Upon motion duly seconded and carried, the President appointed the following standing committees:

"PRACTICE FOR THE PREVENTION OF ACCIDENTS."

(Committee to consist of five members).

J. E. Jopling, Ishpeming, Mich., Chairman; Charles E. Lawrence, Iron Mountain, Mich.; D. E. Sutherland, Ironwood, Mich.; A. M. Gow, Duluth, Minn.; Edward Koepel, Beacon Hill, Mich.

"CARE AND HANDLING OF HOISTING ROPES."

(Committee to consist of five members.)

C. H. Munger, Duluth, Minn., Chairman; O. D. McClure, Ishpeming, Mich.; J. S. Jacka, Crystal Falls, Mich.; Wm. A. Cole, Ironwood, Mich.; Wm J. Richards, Painesdale, Mich.

"PAPERS AND PUBLICATIONS."

(Committee to consist of five members.)

William Kelly, Vulcan, Mich., Chairman; W. H. Johnston, Ishpeming, Mich.; J. M. Bush, Ironwood, Mich.; J. H. Hearding, Duluth, Minn.; F. W. McNair, Houghton, Mich.

"BUREAU OF MINES."

(Committee to consist of three members.)

M. M. Duncan, Ishpeming, Mich., Chairman; J. B. Cooper, Hubbell, Mich.; the Secretary of the Institute.

"MINING METHODS ON MESABI RANGE."

(Committee to consist of three members, to be appointed later.)

Committees to serve until their successors are appointed; each committee to have power to appoint sub-committees as may be deemed necessary.

The following communication from the American Institute of Mining Engineers, relative to an exchange of publications with that society was presented:

"PROPOSED EXCHANGE OF PUBLICATIONS WITH THE LAKE SUPERIOR MINING INSTITUTE."

(1) The Secretary of L. S. M. I. shall transmit to the Secretary of the A. I. M. E., the papers, properly edited which have been passed for publication by the L. S. M. I., together with the necessary illustrations or drawings pertaining thereto; also the manuscript of business proceedings which are usually published in the annual volume of the L. S. M. I.

(2 The A. I. M. E. will print advance copies of such papers for use at the meetings of the L. S. M. I., in number not exceeding the

number of members of the L. S. M. I., and forward them in bulk to the Secretary of the L. S. M. I.

(3) The A. I. M. E. will print the collected papers representing the annual volume of the L. S. M. I. as one of its monthly bulletins, sent regularly to all members of the A. I. M. E., and will print for the L. S. M. I., 750 copies, a number equal to the number of the members of the L. S. M. I. bound in cloth, and the rest bound in paper, uniform with the present published transactions of the L. S. M. I. and including proper title page and the two extra forms containing the business proceedings of the L. S. M. I., but omitting the introductory pages printed for members of the A. I. M. E. These 750 copies will thus be practically identical with the present publication of the L. S. M. I.

(4) Any excess of this annual volume over 240 printed pages of papers, or 32 pages of business proceedings, shall be paid for extra at the rate of $2.00 per page.

(5) The A. I. M. E. will send direct, prepaid, properly wrapped, from these 750 copies, to the entire mailing list of the L. S. M. I. and will ship the remainder in bulk to the Secretary of the L. S. M. I., the cloth bound copies being in proper mailing cases.

(6) The A. I. M. E. does not obligate itself to reproduce the papers of said bulletin in its annual bound volume, but is empowered to so re-publish such of them as may be passed upon favorably by its own publication committee.

(7) The A. I. M. E. will send all members of the L. S. M. I. individually, all other issues of the monthly bulletin of the A. I. M. E.

(8) As compensation for the above, the L. S. M. I. shall pay to the A. I. M. E. $3.00 per annum per member of the L. S. M. I., payable on the basis of the average number to which the regular bulletins of the A. I. M. E. are sent under this agreement, and payable on delivery of the remainder of the special edition of 750 copies to the L. S. M. I.

(9) This agreement shall go into effect as soon as it is accepted by the managing board of the L. S. M. I.

(10) If either Institute desires to discontinue this arrangement, it shall give three months notice thereof to the other party."

Upon motion duly seconded and carried, the President and Secretary were instructed to formulate a reply to the proposal, and mail the same to the American Institute of Mining Engineers. The following is a copy of the letter sent them on July 31st:

"The proposal for the exchange of publications between the American Institute of Mining Engineers and our Institute was submitted to our Council through correspondence between Mr. Richards and Mr. Kelly. On request of Mr. Richards we are addressing our reply to you.

The subject was very carefully considered at a meeting held

Tuesday, July 23rd, and the Council concluded that it could not recommend the acceptance of this proposal by our Institute, mainly because of the additional expenditure which it involves. It was the sense of the meeting that a closer relationship between the two Institutes was very desirable and should be encouraged as far as practicable. The possibility of making arrangements whereby the papers submitted to our Institute could be made available for use by your Institute was discussed, and it was the opinion of our Council that if this could be accomplished without involving serious expense to our Institute that it should be done. If, therefore, your Institute cares to consider further this subject we shall be pleased to assist you.

Our Institute has inaugurated plans for an increased activity among our own members which involves an increase in our expenses, making it especially necessary to conserve our resources.

To Mr. Chas. F. Rand,
29 West 39th St.,
New York City, N. Y."

On motion duly made, seconded and carried the prices of Institute Proceedings are established as follows:

To members for extra copies:

Cloth bound	$1 25
Paper bound	1 00

To non-members:

Cloth bound	$2 50
Paper bound	2 00

The price at which back volumes are furnished to other societies and libraries to be the same as charged to members.

The minutes of the above meeting were submitted to members of the Council not present and has received approval from all

On motion the report was adopted and ordered placed on file.

The following applications for membership, received since the last annual meeting, are approved by the Council:

Andrews, C. E., Superintendent C. & N. W. R'y. Co., Escanaba, Mich.

Atkins, Samuel E., Mining Engineer, 909 Alworth Bldg., Duluth, Minn.

Bayliss, Willard, Superintendent of Iron Mines, Chisholm, Minn.

Byrne, S. E., Manager Paine Webber & Co., Houghton, Mich.

Burt, John H., Mining Captain, Norman Mine, Virginia, Minn.

Boswell, Arthur E., Mining Journal, Marquette, Mich.

Benedict, C. Harry, Metallurgist, Calumet & Hecla Mining Co., Lake Linden, Mich.

Bittchofsky, A. C., Secretary, The Bristol Mining Co., Cleveland, Ohio.

Bowden, Richard, Mining Captain, Trimountain Mining Co., Trimountain, Mich.

Boley, W. E., Mine Superintendent, Baltic, Mich.

Conibear, William, Safety Inspector, The Cleveland-Cliffs Iron Co., Ishpeming, Mich.

Christensen, George L., Professor Mechanical Engineering, Michigan College of Mines, Houghton, Mich.

Clark, Wesley, Mine Superintendent, Copper Falls Mine P. O., Keweenaw Co., Mich.

Cunningham, Mark H., Master Mechanic, Freda, Mich.

Dickerman, Alton L., Mining Engineer, 70 State Street, Boston, Mass.

Daume, Peerless P., Assistant Master Mechanic, Painesdale, Mich.

DesRochers, George E., Mining Engineer, Montreal, Wis.

DeSollar, T. C., Mining Engineer, Quincy Mining Co., Hancock, Mich.

Fellows, Otis D., Jr., Mining Engineer, Redridge, Mich.

Felver, Howard C., Structural Engineer, Houghton, Mich.

Fisher, Henry, Superintendent Calumet & Hecla Mills, Lake Linden, Mich.

Formis, Andre, Superintendent Ojibway Mining Co., Ojibway, Mich.

Gardner, Octave D., Assistant Engineer Calumet & Hecla Mining Co., Calumet, Mich.

Gibson, William M., Assistant Superintendent Calumet & Hecla Mining Co., Calumet, Mich.

Gould, E. P., Salesman, The Lunkenheimer Co., Cincinnati, Ohio.

Hallingby, Ole, Superintendent Laurium Mining Co., and LaSalle Copper Co., Calumet, Mich.

Hocking, Richard O., Mining Engineer, Meriden Iron Co., Hibbing, Minn.

Hayden, George S., Editor of Ishpeming Record, Ishpeming, Mich.

Hingston, E. C., Northern Sales Manager, Bucyrus Co., 707 Alworth Bldg., Duluth, Minn.

Hoatson, Thomas, Mining, Laurium, Mich.

Haller, Frank H., Superintendent Osceola Consolidated Mining Co., Osceola, Mich.

Jaehnig, Frank A., Clerk, Franklin Mining Co., Dunmon, Mich.

Jewett, Norman R., Shift Boss, Red Jacket Shaft, Calumet & Hecla Mining Co., Laurium, Mich.

Jackson, C., Mechanical Engineer, Madison, Wis.

Jolly, John, Underground Superintendent, Painesdale, Mich.

Johnson, R. M., Mine Superintendent, Greenland, Mich.

Kitts, Thomas J., Representative H. W. Johns-Manville Co., Houghton, Mich.

Keast, William J., Mine Accountant, Houghton, Mich.

Knox, John Jr., General Superintendent, Calumet & Hecla Mining Co., Calumet, Mich.

Laist, Alexander, Superintendent Quincy Smelting Works, Hancock, Mich.

Loudenback, Clyde I., Salesman, Crandall Packing Co., 228 West Randolph St., Chicago, Ills.

Leach, Edward J., Manufacturers' Agent, Hancock, Mich.

LaRochelle, Louis, Mining Captain, Houghton, Mich.

Lang, S. S., Superintendent Naumkeag Copper Co., Houghton, Mich.

Mather, S. Livingston, Cleveland-Cliffs Iron Co., Rockefeller Bldg., Cleveland, Ohio.

Mercer, Harry T., Mining Engineer, Painesdale, Mich.

Mullen, Thomas M., Resident Manager Gay & Sturgis, Houghton, Mich.

McIntyre, John E., Engineer of Mines, Nogales, Arizona.

Nickerson, H. F., Salesman, I. E. Swift Co., Houghton, Mich.

Noetzel, Benjamin D., Clerk Trimountain Mining Co., Trimountain, Mich.

Newett, W. H., Editor Iron Ore, Ishpeming, Mich.

Paine, Francis W., Instructor in Geology, M. C. M., Houghton, Mich.

Paine, W. A., President Copper Range Consolidated Co., 82 Devonshire St., Boston, Mass.

Potter, Ocha, Superintendent Superior Copper Co., Houghton, Mich.

Rice, Claude T., Mining Journalist, Mining & Engineering World, Chicago, Ills.

Rehfuss, Louis I., Student, LaCrosse, Wis.

Rankin, William A., Electrical Engineer, Painesdale, Mich.

Rice, John H., President Houghton National Bank, Houghton, Mich.

Raymond, Henry A., Cleveland-Cliffs Iron Co., Rockefeller Bldg., Cleveland, Ohio.

Robertson, James, A. G. F. & P. A., D., S. S. & A. R'y., Houghton, Mich.

Shelden, R. Skiff, Lawyer, Houghton, Mich.

Silver, C. R., Salesman for R. & J. Dick, Ltd., 29 West Lake St., Chicago, Ills.

Strong, Clarence G., Salesman, The Lunkenheimer Co., Cincinnati, Ohio.

Schacht, William H., Mining Engineer, Painesdale, Mich.

Sparks, Benjamin F., Mining Engineer, 205 Ruby St., Houghton, Mich.

Seeber, R. R., Mine Superintendent, Winona, Mich.

Wallace, W. R., Diamond Drill Contractor, Houghton, Mich.

Walker, Elton Willard, Mine Superintendent, Mass, Mich.

Woolf, Percival J., Electrical Engineer, General Electric Co., Minneapolis, Minn.

Word, William W., Manufacturer, 60 Castro St., San Francisco, Cal.

On motion duly carried the Secretary was instructed to cast the ballot for the election of the applicants.

The Auditing Committee presented the following report:

Your Committee appointed to examine the books of the Secretary and Treasurer, beg leave to report that we have carefully examined same and find the receipts and expenditures shown therein to be in accordance with the statements of the Secretary and Treasurer for the fiscal year ending August 26th, 1912.

JOHN. M. WAGNER,
G. S. BARBER,
W. P. CHINN.

On motion the report of the Committee was accepted.

Your Committee on nominations beg leave to submit the following names as Officers of the Institute for terms specified:

For President (one year):
Penticost Mitchell, Duluth, Minn.

For Vice Presidents (two years):
Francis J. Webb, Duluth, Minn.
A. D. Edwards, Atlantic Mine, Mich.
To Fill Vacancy (one year):
William P. Chinn, McKinley, Minn.

For Managers (two years):
G. S. Barber, Bessemer, Mich.
Wm. H. Johnston, Ishpeming, Mich.
Charles H. Baxter, Loretto, Mich.

For Treasurer (one year):
E. W. Hopkins, Commonwealth, Wis.

For Secretary (one year):
A. J. Yungbluth, Ishpeming, Mich.

The Committee further recommended that the salary of the Secretary be $750 for the ensuing year.

M. M. DUNCAN, WILLIAM KELLY,
W. J. WEST, J. B. COOPER,
L. C. BROWN, Committee.

Report of Committee on amendment of rules:

To the President and Members of the Lake Superior Mining Institute:

Your Committee to whom was referred the duty of drafting an amendment to the rules to authorize the Council to expend money through Committees appointed for investigation purposes respectfully recommend the adoption, in the usual way by means of a letter ballot, of the following section to be known as Section XII:

The Council is authorized to appoint from time to time special Committees to consider and report upon, to the Institute through the Council, such subjects as changes in mining laws, safety devices, the securing and editing of papers on mining methods, definition of mining terms, affiliations with other societies, and such other subjects as the Council shall deem it desirable to inquire into, such reports not to be binding on the Institute except action is taken by the Institute in accordance with the rules, and the Council is authorized to expend not exceeding six hundred dollars in any one year to carry out the purpose of this section.

WILLIAM KELLY,
WM. H. JOHNSTON,
F. W. M'NAIR.

On motion the report of the Committee was adopted, subject to the vote of the members by letter ballot as required.

The committee appointed on the resolution on the death of Graham Pope presented the following report:

The Lake Superior Mining Institute has recently suffered the loss of one of its most highly respected and best beloved members in the death of Graham Pope. Becoming early associated with the Institute and being one of its ex-Presidents, he was widely known and admired by its members and his loss in this way becomes a personal one.

In his going Beyond he carries with him the respect and admiration of his associates here in the Institute;

Be it resolved, That this resolution be written in the minutes of the Institute and a copy of the same be given to the members of Mr. Pope's family.

M. M. DUNCAN,
J. B. COOPER,
J. H. HEARDING.

On motion the resolution was adopted by a rising vote.

The following resolution was on motion adopted.

Resolved, That a vote of thanks be tendered to the citizens of the Copper Country, for their cordial reception, to the mining officials for the courtesies extended in the visits to the several plants, to the railroads for the splendid accommodations in their special train service, to the various clubs for the special privileges and entertainments afforded, and to the special Committee of local members and citizens having the matter in charge and making possible such a successful and enjoyable meeting.

F. W. McNair, President of the College of Mines, announced that immediately after the close of the meeting the members would be taken for a short trip through the College buildings where some demonstrations would be witnessed.

SPECIAL PROGRAM.

Pages seventeen to fifty-one comprise a special section giving a description of all the large machinery plants and many of the smaller ones of the copper district. This report was prepared under the supervision of W. L. Carnahan, Boston, Mass., and copyrighted by the Institute. Extra copies may be secured at one dollar each by sending remittance to the Secretary.—(Secretary.)

PAPERS.

Authors of papers printed by the Institute will be furnished twenty-five copies in pamphlet form free of charge. Additional copies will be furnished at actual cost of printing. When sending papers, please specify number desired. Attention is called to the importance of having all papers printed and distributed in advance of the meeting and authors will please send in their copy as early as possible. It is also suggested that drawings to accompany papers be made on a scale to reduce to convenient size.—(Secretary.)

The following is a partial list of those in attendance:

Abeel, Geo. H......Hurley, Wis.
Adams, C. L.......Beacon, Mich.
Allen, R. C........Lansing, Mich.
Armstrong, F. H....Vulcan, Mich.
Armstrong, L. B....Chicago, Ills.
Andrews, C. E...Escanaba, Mich.

Baer, H. L.......Hancock, Mich.
Barber, G. S.....Bessemer, Mich.
Bawden, F. J...Houghton, Mich.
Benedict, C. H..Lake Linden, Mich.
Bengry, W. H.....Palatka, Mich.
Bittchopsky A. C..Cleveland, O.
Blackburn, R. D..Hancock, Mich.
Boley, W. E........Baltic, Mich.
Bolles, F. R....Houghton, Mich.
Bond, Wm.......Ironwood, Mich.
Boswell, A. E..Marquette, Mich.
Bowden, R...Trimountain, Mich.
Bower, L. F....Milwaukee, Wis.
Brady, Samuel..Rockland, Mich.
Brady, M. H....Rockland, Mich.
Brewer, Carl ...Ironwood, Mich.
Brewer, L. C....Ironwood, Mich.
Brigham, E. D......Chicago, Ills.
Broan, J. M.....Houghton, Mich.
Burall, F. P....New York, N. Y.
Burt, John H.....Virginia, Minn.
Bush, J. M......Ironwood, Mich.
Byrne, S. E.....Houghton, Mich.

Campbell, D. H..Iron River, Mich.
Cantillon, W. D....Chicago, Ills.
Carbis, F...Iron Mountain, Mich.
Carnahan, A. L....Boston, Mass.
Carroll, M. J....Houghton, Mich.
Carroll, P......Houghton, Mich.
Carroll, R......Houghton, Mich.
Carroll, J. R....Houghton, Mich.
Caverly, W. D...Houghton, Mich.
Champion, Chas....Beacon, Mich.
Chinn, W. P....McKinley, Minn.
Chinn, R. J.....McKinley, Minn.
Christianson, G. L. Houghton, Mich.
Clarke, E. G....Houghton, Mich.
Clifford, J. M....Escanaba, Mich.

Cook, L. P......Houghton, Mich.
Cole, W. T......Ishpeming, Mich.
Cole, C. D......Ishpeming, Mich.
Cole, T. F........Duluth, Minn.
Conibear, Wm..Ishpeming, Mich.
Conover, A. B.....Chicago, Ills.
Cooper, J. B......Hubbell, Mich.
Cooper, C. H....Houghton, Mich.
Corbett, R. H....Houghton, Mich.
Cory, E. W......Negaunee, Mich.
Creitz, W. J....Houghton, Mich.

Daume, P. P...Painesdale, Mich.
Davis, W. J.......Verona, Mich.
Dee, Jas. R....Houghton, Mich.
Davidson, W. F...............
..........Iron Mountain, Mich.
Dee, Thos. S......Boston, Mass.
Dickerman, A. L...Boston, Mass.
Doty, O. B.....Houghton, Mich.
Douglas, W. C..Houghton, Mich.
Duncan, M. M..Ishpeming, Mich.

Eaton, Lucien..Ishpeming, Mich.
Edwards, A. D................
..........Atlantic Mine, Mich.
Eldridge, P. C..Houghton, Mich.
Estep, H. Cole......Chicago, Ills.

Fay, Joseph....Marquette, Mich.
Fellows, O. D. Jr. Redridge, Mich.
Felver, H. C....Houghton, Mich.
Fesing, H. W....Houghton, Mich.
Fisher, Jas. Jr...Houghton, Mich.
Fishwick, E. T..Milwaukee, Wis.
Flodin, N. P....Marquette, Mich.
Fox, M. J..Iron Mountain, Mich.

Gatch, N. B.......St. Louis, Mo.
Gibson, W. M....Calumet, Mich.
Gish, J. R....Beaver Dam, Wis.
Goodale, G. S...Houghton, Mich.
Goodell, H. S..Painesdale, Mich.
Goodman, F. B Ironwood Mich.
Goodney, S. J. Crystal Falls, Mich.
Goodsell, B. W.....Chicago, Ills.
Gow, A. M........Duluth, Minn.
Grigg, John ....Houghton, Mich.

Hampton, H. C.....Chicago, Ills.
Harris, J. L......Hancock, Mich.
Harris, S. T....Houghton, Mich.
Harrison, G. E....Hibbing, Minn.
Hastings, E....Green Bay, Mich.
Healy, J. T.....Houghton, Mich.
Hearding, J. H....Duluth, Minn.
Hearley, M. T.....Cleveland, O.
Heath, G. L......Hubbell, Mich.
Helmer, C. E....Escanaba, Mich.
Henderson, E...Houghton, Mich.
Heyn, H.......Ishpeming, Mich.
Hingston, E. C....Duluth, Minn.
Hoar, W. B.....Houghton, Mich.
Hodge, W. R....Houghton, Mich.
Holman, J. W......Chicago, Ills.
Holmes, W. J..Washington, D. C.
Holt, H. G.....Ishpeming, Mich.
Hood, O. P......Houghton, Mich.
Hood, B. B.....Houghton, Mich.
Hood, H. P.....Houghton, Mich.
Hopkins, E. W...............
..........Commonwealth, Wis.
Hore, R. E......Houghton, Mich.
Hubbard, L. L..Houghton, Mich.
Hunter, Roy D.....Chicago, Ills.

Imhoff, W. G....Iron River, Mich.

Jackson, C.........Chicago, Ills.
Jewett, N. R.....Laurium, Mich.
Johnston, W. H.Ishpeming, Mich.
Johnstone, O. W..Ironwood, Mich.
Jopling, J. E....Marquette, Mich.
Jory, Wm......Princeton, Mich.

Keast, W. J....Houghton, Mich.
Kelly, Wm........Vulcan, Mich.
Kitts, T. J......Houghton, Mich.
Koepel, E.....Beacon Hill, Mich.
Knox, J. Jr.......Calumet, Mich.

LaCroix, M. F...Ishpeming, Mich.
Lane, A. C..Tufts College, Mass.
Lang, T. H........Duluth, Minn.
Lang, S. S......Houghton, Mich.
Lang, R. B......Houghton, Mich.
LaRochelle, L...Houghton, Mich.
LaRue, W. G.......Duluth, Minn.
Lawton, C. L....Hancock, Mich.

Leach, E. J......Hancock, Mich.
Letz, J. F.......Milwaukee, Wis.
Linsley, W. B....Escanaba, Mich.
Longyear, J. M..Marquette, Mich.
Lynch, Thos. F....Duluth, Minn.

Mann, J. C.....Houghton, Mich.
Moss, C. H......Houghton, Mich.
Martin, T. H......Crosby, Minn.
Mather, S. L........Cleveland, O.
Matt, C. W....Menominee, Mich.
Maurer, R. H....Houghton, Mich.
Mercer, H. T...Painesdale, Mich.
Meuche, A. H....Houghton, Mich.
Miner, A. B....Ishpeming, Mich.
Mitchell, W. A.....Chicago, Ills.
Morris, F. F......Pittsburg, Pa.
Mowatt, N. P......Duluth, Minn.
Mowatt, F. W......Chicago, Ills.
Mullen, Thomas.Houghton, Mich.
McDonald, D. P....Duluth, Minn.
McIntyre, J. E..San Antonio, Tex.
McNair, W. A...Houghton, Mich.
McNair, F. W..Houghton, Mich.
McNamara, T. B.Houghton, Mich.

Newett, W. E..Ishpeming, Mich.
Nichols, W. F.,Houghton, Mich.
Nickerson, H. F.Houghton, Mich.
Noetzel, B. D..Trimountain, Mich.
Nolan, Dan ....Ironwood, Mich.

Oliver, R. L...Point Mills, Mich.
Orbison, T. W....Appleton, Wis.

Paine, W. A.......Boston, Mass.
Paine, F. W....Houghton, Mich.
Parmelee, H. C....Denver, Colo.
Pascoe, P .W....Republic, Mich.
Pearce, E. L....Marquette, Mich.
Potter, Ocha....Houghton, Mich.
Powell, D. W....Marquette, Mich.
Prescott, F. M...Milwaukee, Wis.
Prescott, L. L..Menominee, Mich.
Pryor, John C...Houghton, Mich.
Pryor, R. C.....Houghton, Mich.
Pryor, A. T......Houghton, Mich.

Quigley, G. J........Antigo, Wis.
Quine, J. T.....Ishpeming, Mich.

Raley, R. J........Duluth, Minn.
Rankin, W. A...Painesdale, Mich.
Raymond, H. A....Cleveland, O.
Reeder, J. H.....Houghton, Mich.
Rehfuss, L. I.....LaCrosse, Wis.
Reynolds, Irving.Milwaukee, Wis.
Rice, C. T......Ishpeming, Mich.
Rice, J. H.......Houghton, Mich.
Richards, W. J................
...........Crystal Falls, Mich.
Richards, W. J..Painesdale, Mich.
Ritter, F.....New Britton, Conn.
Robertson, Jas..Houghton, Mich.
Robertson, H. J..Escanaba, Mich.
Rogers, A. F....Ishpeming, Mich.
Rohn, A. W......Baraboo, Wis.
Rough, J. H....Negaunee, Mich.
Rowe, W. C.....Bessemer, Mich.
Rupp, J. A......Ironwood, Mich.
Ryan, E. P....Mexico City, Mex.

Sawyer, A .H....Redridge, Minn.
Scott, H. M........Chicago, Ills.
Schacht, W. H..Painesdale, Mich.
Schubart, Geo. P..Hancock, Mich.
Seaman, E. A....Houghton, Mich.
Seeber, R. R......Winona, Mich.
Shelden, R. S....Houghton, Mich.
Senter, A. W......Hubbell, Mich.
Sherrerd, J. M.......Easton, Pa.
Sherwood, M. W....Chicago, Ills.
Shields, Joseph...Hibbing, Mich.
Shields, I. J.....Houghton, Mich.
Shields, J. W....Houghton, Mich.
Shields, R. H....Houghton, Mich.
Siebenthal, W. A.Republic, Mich.
Siefert, J. A.....Houghton, Mich.
Silver, C. R........Chicago, Ills.
Sisley, L. A........Chicago, Ills.
Smith, W. J......Mohawk, Mich.
Smith, C. G.....Kearsarge, Mich.
Smyth, H. L....Cambridge, Mass.
Sparks, B. F.....Houghton, Mich.
Sperr, F. W.....Houghton, Mich.
Stakel, C. J.....Ishpeming, Mich.
Stanton, J. R....New York, N. Y.
Stevens, C. E....Ironwood, Mich.
Strong, C. G........Chicago, Ills.
Stuart, H. E.....Houghton, Mich.
Sutherland, D. E.Ironwood, Mich.
Swift, H. L......Houghton, Mich.
Swift, P. D......Houghton, Mich.

Tiffts, G. L.....Milwaukee, Wis.
Trevarthen, W. J.Bessemer, Mich.
Tupper, C. A....Houghton, Mich.
Tyler, W. E.....Milwaukee, Wis.

Uren, W. J......Houghton, Mich.

VanOrden, E. L..Houghton, Mich.
Vallett,, B. W....Ironwood, Mich.
Vilas, R. L......Ishpeming, Mich.
Vivian, J. G........Duluth, Minn.

Wagner, John M.Houghton, Mich.
Wallace, W. R..Houghton, Mich.
Walsh, W.......Houghton, Mich.
Ware, J. F............Dayton, O.
Ware, W. F.....Negaunee, Mich.
Warne, D. S....Houghton, Mich.
Wearne, Wm.....Hibbing, Minn.
Webb, F. J.........Duluth, Minn.
Webb, C. J......Houghton, Mich.
Welker, W. F......Ashland, Wis.
Wells, Pearson..Ironwood, Mich.
West, W. J......Hibbing, Minn.
White, E. E....Ishpeming, Mich.
Whiners, W. J..Houghton, Mich.
Whitney, J. H....Oshkosh, Wis.
Whitney, A. B....Oshkosh, Wis.
Williams, P. S....Ramsey, Mich.
Williams, G.....Houghton, Mich.
Wilson, E. B......Scranton, Pa.
Wirtz, R. J......Houghton, Mich.
Witherbee, S. H...Duluth, Minn.
Wright, M. J....Houghton, Mich.
Woolf, P. J........Chicago, Ills.

Yates, W. H....Negaunee, Mich.
Young, H. O....Ishpeming, Mich.
Yungbluth, A. J.Ishpeming, Mich.
Yungbluth, R. O.Ishpeming, Mich.

PAPERS

METHODS OF SAMPLING AT LAKE SUPERIOR IRON MINES.

BY BENEDICT CROWELL, CLEVELAND, OHIO.

RECENT PRACTICE.

In 1904, Mr. E. A. Separk presented a paper to the Institute entitled "Some Aspects of the Analyzing and Grading of the Iron Ores of the Gogebic Range." The methods of sampling, analyzing and grading ores then in use were described in detail. Since that time nothing has been written on the subject, in spite of the fact that conditions have changed greatly, and the new conditions have been met and the new difficulties often overcome by new methods. This lack of information of the present practice may justify the existence of this paper.

The facts brought out by Mr. Separk's paper are worth special attention. The sampling methods were based on the assumption that the sample should contain the same proportion of lump ore, fine ore, and rock (if present) as the ore it represents. The methods described by him were used in all the iron mines of the Lake Superior region, and had proved satisfactory, and had checked very closely with the furnaceman's average.

The impression left on the mind of the reader of this article and discussion was that the sampling methods were correct in principle, and gave good results when applied to large blocks of ore, say from 50,000 tons up to 750,000, but in individual samples of the same ore, at different places, there were discrepancies.

These results on large blocks of ore were, of course, the

average of the results of from 30 to 300 cargo analyses, each cargo containing say 100 cars, and each car being generally sampled as a unit. Each block therefore represented an average of from 300 to 3,000 samples, and results on these large blocks can easily be made to check by a compensation of errors.

In the discussion of this paper, Dr. N. P. Hulst stated that the low phosphorus ore from the Pewabic mine was sampled from the railway cars, first laying a net across the top of the car, and then taking the samples from under the knots of the net. This method was very accurate and there never was any complaint on the part of the buyers of the ore. Mr. T. F. Cole and Mr. J. R. Thompson agreed that the main sampling checked up closely with the lower lake chemists and furnaces "in the long run," that is, the averages checked closely. Some discrepancies were mentioned. For instance, "Managers had experienced different results from samples taken underground and from those taken on the surface, from the same ore." "Samples taken from the top of cars underground holding one ton have differed from those taken from skips holding two tons of the same ore." "The discrepancies seem to occur from an improperly mixed sample, dissimilar methods, the use of impure chemicals, and imperfect manipulation." In short, the mine's average of the seasons shipments of ore checked up closely with lower lake averages. Individual cargoes did not check so well; in fact, they were hardly expected to check, and daily output samples taken at the mines often showed serious discrepancies.

Present Conditions.

At the present time to ship a hundred thousand tons of ore, and to have the average result of the producers' and consumers' chemists check by .1 per cent or .2 per cent in iron is not considered conclusive evidence of good sampling. It is now necessary for each cargo to be taken as a unit and the producers' and consumers' samples of each individual cargo must check closely. Pig iron is now more closely graded than

in 1904; furthermore, it is now bought and sold on specifications that allow such small variations in silicon and manganese that the exact analysis of each cargo of ore must be known. This has resulted in more careful sampling at the consumers' end. But the consumers' analyses and the lower lake chemists' analyses are often received too late to be of value to the furnaceman. Some of the ore from a cargo is often in the furnace before the entire cargo is unloaded. For this reason the mine analysis of each cargo must be used by the furnaceman, and it must be correct.

Under present day conditions, then, sampling ore at the mines has assumed a new importance. Have the sampling methods and practice at the mines kept pace with the new requirements, and with the sampling methods now used at the lower lakes?

At the present time many methods of sampling are in use. Each mine has had its own particular problem and has worked it out in its own way, until today many methods and modifications of methods are in use, some of them good and some of them bad.

PRESENT SAMPLING METHODS.

There are four places underground where samples are generaly taken.

FIRST. *Faces of Ore in Place.* These are generally sampled by the shift bosses, and at many mines no regular method is used for this sampling. Pieces are broken off the face here and there, often by a candle stick. If seams of rock run through the face, little or none of it is put into the sample. Some shift bosses reason that rock is thrown out before the ore is loaded and should, therefore, not be represented in the sample. Undoubtedly such are the orders, but such orders are not always carried out.

SECOND. *Piles of Ore After Blasting Down.* These piles are sampled by the shift bosses in various ways, by moving around the pile taking pieces at intervals, by sizing up the proportion of lump and fine and taking five or six pounds of

ore in such a proportion; or by taking handfuls here and there, mostly fine ore which is the more convenient to pick up. In sampling mixtures of lump and fine ore often times the sampler will not have a hammer with him.

THIRD. *Stopes and Chutes.* A very common practice in sampling ore from a stope is to take the sample from the chute below just before loading a car. This generally means taking handfuls of ore, mostly fines, for seldom is a hammer used If the samples are taken in the stope, it is by picking from the piles of broken ore, or in some cases, breaking samples off the faces of ore. In any of these conditions it is seldom that a correct proportion of lump and fine is obtained, and sometimes no attempt is made to keep such a proportion in the sample. When the lump and fine ore are approximately of the same analysis, this does not, of course, make much difference. But there are many ores in which the lump contains appreciably more iron than the fine and vice versa.

FOURTH. *Skip and Mine Car Samples.* These samples are generally taken by the skip tender and are supposed to be representative of the ore hoisted for a day, or a shift. They are, of course, continuous samples. A certain amount, varying in different mines from a pinch to a handful, is taken from the top of each car or skip. The ore so taken is generally thrown into a box. The contents of this box are mixed at the end of the day, or shift, and the sample represents the ore hoisted during this period. It is no wonder that these samples are unsatisfactory. The skip tender usually does this work. He is a skip tender first and always and, quite naturally, taking the sample is work to which he attaches small importance. He has been trained to load the skip quickly and is blamed for any delay. If he fails to sample one skip, he knows that he can take two samples from the next skip, and all skips look alike to him. If he fails to sample ten skips, he can make up for it by taking twice as much from the next ten. I heard of a case where a skip tender began well, but gradually neglected his sampling, till finally he filled

his box from the last skip, and for several weeks the grades of ore hoisted were made up from the samples taken this way, and when his sampling method was finally discovered, he was quite proud of his own ingenuity, and could not understand why it was wrong. My belief is that skip samples as taken today, do not represent more than one-half of the ore hoisted, on the average. This neglect is not the worst objection to skip sampling, however, and can be practically overcome by stationing a sampler at the skip. How is it possible to take a continuous sample such as this, and preserve the proper amount of lump and fine ore in the sample? The common practice is for the skip tender to take a small amount of ore from the top of each skip, or mine car. He takes it with his hand, not often using a trowel. I have never seen a hammer in use. The result is the sample contains nothing but fine ore, all lumps larger than one inch in size are ignored, since anything larger than this must be broken. Now, suppose the ore contains 20 per cent of lump and 80 per cent of fine, how can this proportion be preserved in the sample? Several plans have been proposed, but they all take time and involve delaying the skip, and it will be shown later that there is a better place for sampling the output, where no delay is probable.

Pocket Samples—These samples are usually taken at the chute on surface as the ore is run from the pockets into the railway cars. They are generally taken by the lander, whose main duty is to load the cars. The lander usually lets each skip load run through the pocket into the car, without filling the pocket, and holds a shovel in the stream of ore long enough for the shovel to fill. In this way he gets a shovelful of ore to represent the skip load. He then takes a part of the shovelful and places it in a box. At the end of the shift he has a box full of ore to represent the hoisting during that shift. Sampling such as this is open to the same objections as skip sampling, and it is nearly impossible to get proper percentage of lump and fine into the sample.

If the ore is crushed, the skip dumps its load onto a grizzly, the finest passing through. The over size goes over the grizzly to the crusher and passes through the pocket into the car after the fines. In this case the lander often takes a shovelful of fines, and a second shovelful of the crushed lumps, placing a part of each shovelful in the sample box. Since the fines and crushed lumps come to him separately, he necessarily gets some of the lump ore into his sample. How can he get the proportionate amounts right? Often the result is that he takes equal amounts of lump and fine, while the proportions may vary enormously. The box of ore containing the sample is shipped to the laboratory to represent the ore loaded during that shift. If the sample is too large, it is reduced in size by quartering, or some other device to accomplish the same result. At some mines the sample is taken by the lander after the ore is dumped into the car. A small amount is taken after each skip load is dumped, placed in a box and treated as above described.

Theoretically, this method is a great improvement over the other method described, but practically it requires much more work of the lander, and can only be used where loading is slow; even then, the lander probably neglects to sample many skip loads and a better plan would be to sample the surface of the car after it is loaded. This sampling is usually done with the hands. It is unusual to find a scoop being used, and still more unusual to see a hammer used to break up the lumps.

Car Sampling—Car samples are taken from the surface of each railway car after the car is loaded and before it is hauled away from the mine. At some mines this sample is taken by the lander. He samples the car as best he can, when his other duties will permit. He is generally supposed to use his judgment as to the proper proportion of fine and lump ores to put into his sample. In other mines this car sampling is given the attention of trained samplers. These samplers show all stages of proficiency and get varying results. Some use

no tools. Some of them use a scoop or trowel, and a few use both scoop and hammer. Some measure the distances properly, others guess the distance, and still others take the samples haphazard. These samples usually weigh from five to ten pounds per car. They are sent to the laboratory in boxes, each box marked with its proper car number. These samples are sometimes analyzed separately, and sometimes grouped into units of five or ten cars for analysis.

Cargo Sampling—Cargoes of ore are now considered units, and the main object of sampling, at the mine, dock and furnace, is to get a correct sample of each cargo of ore. Underground samples and daily output samples are valuable aids to the miner, but are of secondary importance. Sampling the ore in the vessel while loading has not been successful. The nearly universal practice of Lake Superior mines is now to sample the railway cars at the mine, in groups of from one to ten cars, and with these car analyses, and a list of the car numbers loaded into each cargo, to calculate the analysis of the cargo. This method has proved fairly satisfactory in the past, and will always prove satisfactory, if the car sampling is well done, and the sample truly represents each carload. If the car samples are incorrect, the cargo analysis must be incorrect.

ADVANTAGES OF UNIFORM METHODS OF SAMPLING.

There are two reasons why accuracy in sampling is essential:

First. That the seller of the ore may receive his just due.

Second. That the user may go about the reduction of the ore to pig iron in an intelligent manner.

The calculation of a blast furnace burden is dependent on the analysis of the material to be used in the furnace. If, now, the analysis of a certain ore gives one result at the mine, another result at the lower lakes, and still another result at the furnace, the furnace manager must necessarily be uncertain of just what the ore actually contains, and allow a wide enough variation in his calculation to take care of the dif-

ference. If, in the final analysis, that of the furnace yield, the results show lower iron than was paid for, the seller at once receives a complaint. If the yield is higher, possibly not much is said, but the seller has failed to receive his just due.

It is very desirable and essential that the analysis be as near correct as possible. Analytical work has been standardized until, at present, the various constituents of an iron ore can be determined accurately, at least within narrow limits of errors. This has been accomplished only after careful study and the adoption of standard methods of analysis. If the sample is not representative, then the analytical work, however accurate it may be, is of no practical value. The thing to do then is to standardize the sampling so that the sample will be representative. It is obvious that no method can be devised which can be used at mine, lower lakes, and furnace, but I believe a particular method can be found, applicable to each of these places. This has been done at the lower lakes, is slowly being brought about at the mines, and is being agitated at the furnace.

Objections to Present Methods.

Present methods of sampling at the mines are unsatisfactory in that the results are not wholly reliable. In many cases the methods themselves are wrong in principle, and the details of their execution are not properly carried out. The great absence of trained samplers is especially noticeable.

In many cases underground samples are haphazard affairs. Skip samples, lander's samples, and other forms of continuous output samples, are not altogether successful. Many mines take two continuous samples of the daily output, one underground, the skip sample; and the other at the surface, the lander's sample. These samples represent the same ore, and are taken at practically the same time. Compare their results. The figures for two days' output seldom agree, differences sometimes amounting to 3 or 4 per cent of iron. They can't both be right. The fact is, they are both generally wrong. Compare the results with the samples taken from the

railway cars after loading, and you get three sets of figures all representing the same ore, generally all different. And these conditions are found today at some of the best managed mines on Lake Superior. These three samples are taken daily, and represent the same ore. They should check closely. Now if they do check closely, why not omit two of these samples, and depend on one sample, thus relieving the laboratory of a lot of useless work. And if the samples do not check, what value are they? The fact that the three samples of the same ore do not check proves that at least two of the samples are wrong, and what information can be had from three analysis of the same ore, all different? After carefully watching these output samples at many mines, I believe that the car samples are usually the most reliable of the three and that the other samples should be abolished.

Proposed Uniform Methods.

To meet the demands and necessities of the present conditions, an improvement is necessary. The only logical solution is standardization of sampling methods. It is true that local conditions vary, making a standard method difficult to apply in all cases. I believe, however, that the methods may be made flexible enough to fit practically all conditions. This has been accomplished at lower lake ports. Here samples are taken while the ore is being unloaded, under extremely variable conditions. The Standard Method of Sampling, introduced in 1907, and gradually improved in details, is more successful each year. A description of this method as now applied follows:

STANDARD METHOD FOR SAMPLING CARGOES OF IRON ORE AT THE LOWER LAKE PORTS.

A standard sample shall be taken from all cargoes, the weight of the sample varying with the size of the cargo. The sample shall be taken with a galvanized iron scoop 3½ inches wide, 2¼ inches long, and 1¼ inches deep, the handle 8 inches long; and with a hammer 12 inches long (the scoop holds approximately ½ pound).

It shall be the aim to take an equal bulk of ore from every point selected. When a lump is encountered, a portion shall be

broken off equal in bulk to a scoopful of soft ore. In sampling cargoes no sample shall be taken from the original outside surface on account of the presence of foreign matter and an undue proportion of fines.

If the surface to be sampled has been exposed to rain or sun long enough to materially alter the amount of moisture present, the sample shall be taken about three inches below the surface.

In order to keep the size of samples within reasonable bounds and to gauge the size to the size of the boat, the sampler shall on cargoes up to 3,500 tons, begin sampling at a convenient point, at the bottom of the face of the ore and shall take one standard scoopful every two scoop lengths up the face of the ore to the top, and then shall move four scoop lengths to one side of the starting point before again sampling vertically. He shall continue in this manner keeping the above distances around the face of the ore to the place of beginning.

On cargoes from 3,500 to 6,000 tons, he shall use the two scoop lengths for vertical distances up the face of the ore, but move six scoop lengths horizontally.

On cargoes over 6,000 tons, he shall use the two scoop lengths for vertical distances up the face of the ore, but move eight scoop lengths horizontally.

In cases of split cargoes, horizontal spacing to be according to the tonnage of each individual ore as specified in the preceding paragraphs.

At convenient stages of unloading, the sample shall be mixed and quartered. This must be done each time exactly alike, by breaking down one-half inch, mixing and quartering twice, thus preserving the proper proportion of the whole sample.

If in the final quartering, the last two quarters exceed a can full, the ore shall be quartered again and one quarter rejected.

The sample may be quartered on the vessel, or may be taken to some other place suitable for the purpose. Samples must be shipped in standard cans.

Sampling Soft Ore—The sampler shall enter any hatch and begin sampling when the unloading machines have exposed five or six feet of the face.

The sampler shall then enter the next hatch working, and proceed to sample in the same way, and so continue in every working hatch.

The sampler shall then begin over again in the hatch in which he first started and continue the sampling in all the working hatches, provided there has been sufficient ore removed in such hatches since the faces were sampled to expose fresh ore.

The sampler shall continue this method of sampling in each hatch worked, until there is less than one-tenth of the ore left.

In sampling horizontal surfaces, as in boats where scrapers are

used, the sampler shall sample every two scoop lengths lengthwise of the boat, the spaces between the lines of sampling to be 4-6-8 scoop lengths according to the tonnage as described before.

Sampling of Hard Ore—In sampling hard lump ore the sampler shall begin sampling, and use the same spacing as defined for soft ore, using hammer lengths, instead of scoop lengths. At each point sampled he shall take lump or fine ore equal to one cubic inch. In taking this cubic inch the sampler shall take an average from the lump ore from which the cubic inch is broken.

Moisture Sample—The moisture sample shall be taken from the standard sample in the following manner:

When as many cans of ore have been filled as the stage of unloading will permit, the lump ore shall be broken up quickly and the entire amount thoroughly mixed and flattened out into a circular pile. The pile shall then be marked into quarters and one of the quarters divided into two parts by a radial line from the center to the outside of the pile. The whole of one of the half quarters so marked off is to be placed at once in a tightly covered receptacle to be a portion of the moisture sample. The other half of the quarter together with the opposite quarter are to be rejected.

The remainder of the pile is to be thoroughly mixed and flattened, and two opposite quarters rejected. The remainder of the ore is to be put one side for a portion of the standard sample.

By this method of procedure, one-eighth of the entire sample taken will go into the moisture sample.

At the end of the sampling the accumulated moisture sample is to be taken out of the tightly covered receptacle, quickly mixed and flattened out, divided into eighths and enough eighths taken to fill the standard moisture can.

In case of hard ores or small tonnages of soft ore, the proportion set aside for the moisture sample shall be increased so that at least one can of ore shall remain for the final moisture sample.

The moisture determination is made by drying the whole of the final moisture sample at a temperature not over 212 degrees F. until there is no loss in weight. The loss of weight divided by the amount of ore taken, will give the percentage of moisture.

The methods suggested for the use of the mines, as outlined below, contain no new principles. In fact, they are merely a return to the old sound principles from which samplers in recent years have sometimes strayed.

UNDERGROUND SAMPLING.

Face of Ore—The ore should be sampled in a vertical face, if possible. The broken ore may be caught in a powder box or on a canvas spread on the ground so that it will catch all

of the ore broken from the face. If the ore is hard a hammer and moil, or gad, should be used to break it. A sampling pick is objectionable, since the point of a pick always tends to follow the softer parts of the ore, thus getting an undue amount of softer material in the sample.

The sample should consist of all the ore broken or chipped from a groove cut straight across the formation, at right angles, or nearly so, to the dip. The groove should not be less than one inch deep and two inches wide, and should extend clear across the face of the ore sampled. Several of these grooves may be necessary. This depends upon the size of the face, and whether the ore is mixed or fairly clean. Experience alone will demonstrate the number of cuts necessary for a given size of face in any mine.

If the ore is soft and uniform, use a sampling pick. The groove should be two inches deep and four inches wide. If the ore is mixed, a hard ore mixed with soft rock, or soft ore containing layers of hard lean ore or rock, make a wider cut, using moil and hammer, being careful to keep the groove uniform in size.

Broken Ore—If the ore has been blasted and is lying at the bottom of the drift, or stope, begin at any convenient spot on one side of the pile and draw a line across the middle of the pile. Then from either end of the pile measure one hammer length along the line, at this point take one scoopful of ore, if fine ore is found at this point. If lump ore is encountered here, break off enough lumps to equal in bulk one scoopful. Then measure another hammer length along the line, and take an equal bulk of whatever is encountered at that point, and so on across the pile. This will usually give a fair sample of a uniform ore. If the ore is mixed and requires a larger sample, draw two or more parallel lines across the pile and sample as above described. The more variable the pile the more difficult it is to sample and the larger the sample should be. As soon as the sample is taken, the lumps should be broken down to about one-half inch in size.

Then mix thoroughly and quarter, rejecting the two opposite quarters. The sample should be reduced to 8 to 10 pounds in size, by alternate crushing and quartering.

Sampling of Output—The object of all sampling done by the skip tenders, landers, etc., is to get a fair sample of the output of the ore during a unit of time, usually a day or a shift. These samples are continuous, that is, a small quantity is taken from every mine car, or skip, and these portions are mixed to represent the output during the time specified.

The difficulties of getting a good sample of this kind have already been described, together with the conflicting results obtained. I suggest that these continuous samples be omitted entirely, and the daily output be gauged by samples of the railway cars, just after loading. Railway cars just after loading, whether from shaft or from stockpile, give a good surface of ore from which to sample. The surface contains about the same percentage of lump and fine ores as exist below the surface. Moreover, the surface is large as compared to the tonnage of ore in the car. There is usually ample time for getting the sample, so the sampler is not hurried, and finally from car samples can be made up composite samples to represent the day's hoist, the shipments from stockpile, and the cargo as shipped. I believe that all energy should be concentrated on getting a good sample from each car.

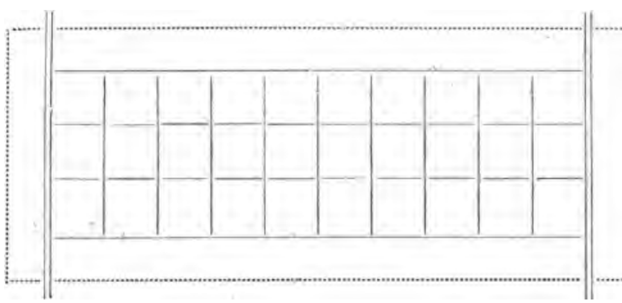
Each sample is marked with the number of the car and crushed to go through a four mesh sieve and mixed. A small part is then taken from each sample representing a car loaded at the shaft, and this composite sample represents the day's hoist. A small portion is taken from each sample representing a car loaded from stockpile, and this composite sample represents the ore loaded from stockpile on that day.

The remainder of each car sample can be filed away until the dock reports the cars which made up a particular cargo, and the samples representing those cars can then be mixed, the resulting composite sample to represent the cargo.

These car samples may be analyzed separately, or bunched

in groups of five or ten for analysis. Using these car samples as units, the scheme best adapted to the needs of each mine may be used, and all other methods of sampling the output may be dispensed with.

Car Sampling—To sample the surface of a car of ore, I believe that the rope net system is the best. The sketch below shows the net stretched over the surface of the car.



There are usually thirty-two knots in the net. The same amount of ore should be taken from under each knot, taking lump or fine ore, whichever occurs under each knot. This method is slow and is not in favor on that account. The following method is more flexible and is quicker, and has given good results wherever used. The principle is the same in both methods.

Begin at one end of a car in the center, measure two hammer lengths along center line parallel to sides. At this point take a scoopful, if fine ore, or an equal bulk if lump ore. Measure the same distances and take same quantities in the same way until the other end of car is reached. Then repeat the operation on a line one-half way from center line and side, and then repeat again on line one-half way from center line and other side. For ordinary ores have hammer of such length that this proceeding will take ore from five points in each line, or fifteen points per car. For difficult ores, shorten the length of the hammer so as to touch eight or ten points in each line, giving twenty-four or thirty points to each car.

A sketch of the surface of a car will make this method clear.

C	....p....p....p....p....p....	D
B	....p....p....p....p....p....	A
F	....p....p....p....p....p....	E

PP—Points from which ore is taken.
AB—Line of first sampling.
CD—Line of second sampling.
EF—Line of third sampling.

If the sample is too large for convenience break the lump down to one-half inch in size, mix and quarter down to eight or ten pounds.

These methods give a fair sample where the surface of the ore is representative of the ore in the car. In most cases this may be safely assumed. Occasionally this is not so. If the ore is lumpy and the car is loaded slowly from a chute, there is sometimes a tendency for the lump to segregate on the further side of the car. It may be necessary to shift the three lines along which the sample is taken over to one side of the car, as shown below.

C	....p....p....p....p....p....	D
B	....p....p....p....p....p....	A
F	....p....p....p....p....p....	E

In this way the difficulty due to segregation may be overcome. When the location of these sampling lines is once fixed, however, it should not be changed and all cars should be sampled alike.

Experienced samplers should always be used, if possible, men who have no other duties.

Mechanical Sampling Devices.

It must not be assumed that the method of sampling output above described, is in any way superior to a mechanical sampling device such as has been installed at the American mine. At this mine the ore is carried from the crusher to the cars by belt conveyors. At regular intervals of time, the entire stream of crushed ore is diverted from the car to the sample box automatically. When the loading of the car is completed, the box contains a sample representing the car. It is hard to see how this sample could be improved. The conditions necessary to the successful use of mechanical sampling devices are found in comparatively few mines, and their general use seems unlikely.

Stockpiling.

The above methods cover the ground for all mines that are shipping to the docks. When a mine is stockpiling its output, the problem is different.

In this case, the stockpile may be sampled at regular intervals, say twice or three times a day. However, the objections to this scheme are numerous. The difficulties of sampling a stockpile are well known, and weather conditions are very bad during the usual stockpile season, that is, in the winter months. In some cases, it will be found better to take a continuous sample of the ore, taking the sample from the top of the cars that carry the ore from the shaft to the stockpile. An equal bulk of ore should be taken from predetermined places on the surface of each car load, taking fine or lump, whichever occurs at these points.

Sampling a Stockpile—In sampling a stockpile of ore the first point to be decided is, does the surface of the stockpile truly represent the ore in the pile. If it does not, the pile must be trenched, or otherwise opened up. Just how much the pile should be opened up is a troublesome question. Work such as this takes time and money, the amount being determined by the importance of the sample and how soon the information is needed. If the stockpile is to be bought or sold,

the exact analysis is of great importance, and justifies the expenditure of considerable money. If the information wanted is only rough approximation of the truth, a quick surface sample will often answer.

In general, if a steam shovel is working on the stockpile, or any part of it, a cross section of the pile more or less complete, will be exposed, and a sample taken in the cut or face on which the shovel is working will be more reliable than a surface sample of the pile.

In stockpiling ore, there is usually more or less separation of the lump, some of which tends to roll down the outside of the pile, thus forming a ring of lump ore around the base of the pile, if the pile is cone-shaped. This free natural spreading of the ore is seldom found, however, most stockpiles being formed by dumping the ore from cars on a trestle and its natural movement is more or less restricted by bulkheads.

When a stockpile is to be sampled, therefore, the sampling should be planned by the best sampling authorities available. In case of the average mine, this will be the chief chemist, or whoever is held responsible for the sampling. This man, fully informed as to points above mentioned, can then give instructions to the sampler as to the method to be pursued. The ordinary ore sampler is seldom capable of deciding these matters.

The method having been decided, the sampler should work along the lines laid out, measuring distances, and taking an equal bulk of whatever he finds at each of the sampling points, as determined by his measurement.

CONCLUSION.

To Summarize—To improve sampling I would urge as follows:

1. Centralize the responsibility by placing all sampling under the charge of one intelligent man, preferably the chief chemist. Then see that his routine laboratory work does not occupy all of his time. Sampling must be watched every day.

2. Have all sampling done by trained samplers, except in certain cases where the expense would be prohibitive.

3. Concentrate all energy on sampling the cars, and abolish all other forms of sampling the output, thus relieving the laboratory of a large amount of work.

4. Adopt a uniform method of sampling, along the general lines above described, but best adapted to the requirements and conditions of each individual mine.

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	Vol.	Page.
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SYSTEM OF SAFETY INSPECTION OF THE CLEVELAND-CLIFFS IRON CO.

BY WILLIAM CONIBEAR, ISHPEMING, MICH.

Conservation of human life is the slogan of the industrial world of today. Recognizing that it is of prime importance to prevent injury, not alone from a humanitarian standpoint but also because it pays as a business proposition, the federal government through the Bureau of Mines, the great railway systems, nearly every coal mining company, many of the iron mining companies and large and small industries such as factories, foundries, mills, furnaces, etc., are making special efforts to preserve life and abate suffering. Interest in this problem is increasing rapidly and it promises to become a national movement that will provide ultimately the maximum safety and efficiency for all "men who are engaged in satisfying the wants of others as a means of providing their own." It needs but a comparison of the loss of life in the industrial world of European countries and the United States to prove that this movement is one that demands our serious consideration and is worthy of our best effort.

That members of the Lake Superior Mining Institute have been cognizant of the high fatality rate in our industry is evident by the number of papers published by the Institute, which treat of this subject.* The presidential address of 17 years ago is an elaborate review of the causes of fatal accidents and invites the co-operation of state, employer and employe to endeavor to check the "unnecessary loss of life." The Presidents for the years 1898 and 1909, in their addresses, discussed the ethical aspect of this subject by calling attention

*See bibliography appended.

to the compensation that should be given to injured workers and their dependents. The recent enactment of a Compensation Law by the Michigan State Legislature is the result of a movement which was initiated by a member of the Institute. The specific hazards that characterize the industry, such as the handling of high explosives, shaft openings, cage riding. etc., have been discussed by those who have been well qualified to offer recommendations for providing greater safety.

The economic phase of mining, however, has been paramount and perhaps there has not been the amount of attention devoted to safety that it is justly entitled to. As a consequence the fatality rate has not shown a marked reduction nor does it compare favorably with the fatality rate in other mining districts of the country. The reports of county mine inspectors give detailed descriptions of accidents but they fail to discuss their causes and means for their prevention. It must be acknowledged that progress has been made in improving conditions in and about the mines, but much remains to be done to reduce the high fatality rate of the past to at least one that, by comparison with the other important mining countries, will not reflect seriously upon our mining methods and conditions.

Because the state failed to meet the demand for more adequate mine inspection but left it to each mining company to solve the problem, The Cleveland-Cliffs Iron Company organized what has been designated its Safety Inspection Department. This department has been in existence about 18 months, a period of time inadequate to permit a comparison of results. The statement has been made that one-half of the accidents in our country are preventable*, but if their prevention were an easy matter surely the number would have long since been reduced. While the reduction of accidents is not only possible but imperative as well, it will require much careful study, backed by an intelligent and systematic effort

*American Red Cross Text Book on First Aid, Pages XI and XII.

which must have the co-operation of employer and employe to accomplish this result.

RULES AND REGULATIONS.*

A preventable accident is caused either by the carelessness or ignorance of an employe or by the failure of an employer to provide adequate safety devices. As one means of insuring safe working conditions and for the purpose of assisting its employes to exercise safety precautions, The Cleveland-Cliffs Iron Company has published a book of rules and regulations. This book has been issued in two forms, one for superintendents and foremen and one for all other employes. The first is printed in English and is in two parts; one includes a foreword from the president of the company and the rules and regulations for superintendents and foremen, and the other part includes the general rules and regulations. The second form is printed in English, Finnish and Italian, which are the several languages spoken by a large majority of the company's employes, and includes a foreword and general rules and regulations. Every employe receives a book and his signature for the same and agreement to study and live up to it is filed at the central office of the company. Within a reasonable time he must satisfy his foreman or boss that he is familiar with the rules of the company. The first violation may be sufficient cause for a suspension of six days or longer and a second is usually followed by dismissal from the company's service.

It has always been a rule of the company to post in conspicuous places copies of the bell signals, the special rules which govern the use of explosives, and the duties of hoisting engineers and motormen. It is not necessary to add that this practice is still in force.

*The rules adopted by the Company are based upon the report on Uniform Mining Laws made on September 1st, 1910, by a committee consisting of Walter Benton Ingalls, Chairman, J. Parke Channing, James Douglas, James R. Finlay and John Hays Hammond, to the American Mining Congress, American Institute of Mining Engineers, and the Mining and Metallurgical Society of America, with such modifications and additions as found necessary for their mines.

SAFETY INSPECTOR.

The safety department is in charge of a safety inspector whose duty it is to inspect all places where men are working or through which they are obliged to travel. His reports are submitted to the company's agent, to whom he is responsible for a conscientious performance of his duties. Accompanying this paper is an inspection blank, which outlines the kind of an inspection that is necessary in order to make the report. The blank is subject to amendment, and the inspector is at liberty and is expected to report all dangerous conditions that come under his observation and which may not be brought out by the questions embodied in the blank. A proposed inspection of a mine is not announced and therefore no opportunity is given the men in charge to make changes that may be necessary to meet the requirements of the rules. As failure on the part of the inspector to report impartially all violations of rules or dangerous conditions would frustrate the very object of his official existence, this method of inspection is one that stimulates a foreman to keep the mine in such a condition at all times as to comply with the safety rules and keeps him alert for unsafe or unguarded places. The inspection form and the examinations by the foremen's and workmen's committees give the inspector no opportunity to offer a plausible excuse if he has been derelict in the discharge of his duties.

FOREMEN'S SAFETY COMMITTEE.

Safety inspections are made periodically by a committee of mine foremen, consisting of three members, who are selected from different mines of the company. The committee is instructed directly by the agent and each member is furnished a copy of the questions, which form the inspection. The safety inspector accompanies the committee on its tour of inspection, in the capacity of secretary. He has no voice in the decisions, which the committee may reach, but has to record them and incorporate them in a report, which is read very carefully by each member of the committee before he

attaches his signature thereto and permits it to be transmitted to the agent. The committee makes one inspection of all mines and is then discharged.

An inspection of this kind may be the means of suggesting points of possible danger that might otherwise pass unobserved by a safety inspector, and it assures to the company the experience and judgment of men who should be best qualified to recognize dangerous conditions and unguarded appliances. During the brief life of the safety department of The Cleveland-Cliffs Iron Company the members of the foremen's committee, without exception, have esteemed it a special favor to make a safety inspection and have gladly accepted the opportunity of being a factor in providing greater safety for their fellow workmen as well as for themselves. It is a trait of human nature that whenever rules and regulations, which affect a body of men as individuals and as a whole, are formulated, to evince a curiosity as to how the "other fellow" is obeying them. And when the members of a committee learn that the safety precautions, enforced in the mines at which they are employed, are but a part of a system in vogue at all mines, they conclude the movement is one that has come to stay and that it behooves each one, individually, to fall in line and keep up with the "other fellow."

WORKMEN'S SAFETY COMMITTEE.

A method of accident prevention is not complete unless it aims to win the confidence and co-operation of the workmen, and with this object in view the company has its workmen's safety committees. These committees work in conjunction with the safety inspector and the procedure of preparing a report is similar to that of a foremen's committee with one essential difference and that is that each mine is inspected by a committee of its own workmen, whereas one foremen's committee makes an inspection of all mines. Workmen of different nationalities are chosen, and each successive inspection means three additional employes at each mine, who have given the working conditions of that mine their approval or else

have offered recommendations to lessen danger. There are many foreigners employed in the mining industry, who speak little or no English, and it is interesting to observe the curiosity which they frequently exhibit when a fellow countryman happens to be a member of the safety committee. When the mission of the committee has been duly explained then the workmen must satisfy the members of the committee that they are complying with the rules and regulations by producing cap crimpers and skewers, by showing, that detonators are stored apart from dynamite, etc. Thus the workmen are brought to understand the safety precautions which the company is exercising and habits of indifference and recklessness are supplanted by habits of interest and caution.

CENTRAL SAFETY COMMITTEE.

The central safety committee is composed of the company's mine superintendents of Marquette County, the head mining captain, the master mechanic, the assistant auditor, the secretary of the pension department and the safety inspector, with the agent as a member ex-officio. This committee meets monthly, and acts as a legislative body on all safety recommendations which may have been offered by the foremen's and workmen's committees and the safety inspector and to the application of which objection has been raised. A majority vote of the committee is recorded, and as such is communicated to the agent as the recommendation of the central safety committee. As soon as it is approved by the agent it must be enforced. If a recommendation is considered of sufficient importance to justify a special rule that shall have general application at all mines it is accepted as such and is accordingly classified under the heading "New Rules." In short, all matters of safety, the expediency of which give rise to difference of opinions, are referred to the central safety committee for decision. The committee receives reports of all accidents, whether fatal, serious or slight, which have occurred during the previous month, and classifies them by causes, using for its classification a key which was obtained

from Mr. R. J. Young, Manager, Department of Safety and Relief of the Illinois Steel Company and which is herewith printed by his permission. Preventable accidents are carefully considered and recommendations are suggested for preventing their recurrence.

FIRST AID TO THE INJURED.

It is hardly necessary to call attention to the value of first aid to the injured, but its importance cannot be overestimated. Even if the day of preventable accidents were a thing of the past, the non-preventable accidents are inevitable. Especially is this true of a trade as hazardous as iron ore mining, with its deep mines and constantly increasing mechanical appliances. In certain mines in Pennsylvania first aid instruction and that alone has resulted in reducing sick and death benefits 50 per cent.* A simple fracture may become a compound fracture if the patient is not properly handled, and hence an accident which requires a few weeks of comparatively painless healing becomes one that lengthens into a much longer time of confinement, with considerable danger of death if wound infection develops. This is an illustration of but one kind of injury which frequently occurs in mining and is cited merely to emphasize the value of having workmen around the mines, who have a practical knowledge of the essentials of first aid to the injured. The Cleveland-Cliffs Iron Company's first aid organization and training was started under the instructions of an employe of the Bureau of Mines, whose services were obtained for a month. During this time a number of employes became competent to continue the work. A corps of five members, consisting of both surface and underground employes, is under training at each mine. Each member is furnished with a copy of the American Red Cross "Text Book on First Aid." A drill is held regularly each month and individual and team work are the features. Demonstrations in which all teams participate are held as soon as the teams under training become proficient. When the mem-

*Red Cross Text Book on First Aid, Page XII.

bers of a corps become competent to administer first aid treatment, a new team is selected and each member of the retiring team is given a certificate of competence to administer first aid to the injured, with the stipulation that he must undergo additional training every sx months.

MINE RESCUE STATIONS.

The Cleveland-Cliffs Iron Company has established three mine rescue stations which are equipped with oxygen helmets, electric portable lamps, pulmotors and quantities of supplies. A man is delegated to look after each station and train men at all mines which are assigned to his station. Employes, who are familiar with the working places of the mine in which they work, such as foremen, timbermen, trackmen, etc., are chosen as members of the fire corps. The training consists in explaining the apparatus to the men, putting it on them, sending them in stopes or drifts after a blast, building brattices, carrying men, and such other work as might be expected in an actual case of fire. Each corps is drilled at least once a month and a record is forwarded to the central office of the company. A monthly report of the condition of the equipment at each station is made out by the foreman in charge and is also sent to the central office.

The pulmotor, a unique mechanical contrivance, is an automatic resuscitating apparatus for restoring a person overcome by gas. It has been used very successfully in the large cities and is an essential part of a mine rescue station. In addition to a pulmotor at each of its rescue stations, the company has one at those mines which are located several miles distant from a station. The probability of underground employes being overcome by gas resulting from the use of dynamite is more constant than the probability of a mine fire. Accidents of this nature occur very frequently and a successful application of this apparatus upon a patient is more efficacious when there is little delay in administering the treatment.

Classification of Mine Accidents.

I TRADE RISKS (INCIDENTAL AND NON-PREVENTABLE).

II NEGLIGENCE OF COMPANY.

1. Failure to use Safety Devices Provided.
2. Failure to use Proper Tools or Appliances Provided.
3. Violation of Rules.
4. Improper Act or Selection of Improper Method of Doing Work. (By Foreman).
5. Failure to Instruct Men as to Method of Doing Work and Hazards Incident Thereto.
6. Failure to Provide Safety Devices.
7. Failure to Provide Proper Tools, Appliance or Place to Work.

III NEGLIGENCE OF WORKMEN:

(A) INJURED MEN:
1. Failed to use Safety Devices Provided.
2. Failed to use Proper Tools or Appliances Provided.
3. Violation of Rules.
4. Improper Act or Selection of Improper Method of Doing Work. (By Workman).

(B) OTHER WORKMEN:
1. Failed to use Safety Devices Provided.
2. Failed to use Proper Tools or Appliances Provided.
3. Violation of Rules.
4. Improper Act or Selection of Improper Method of Doing Work. (By Workman).

THE CLEVELAND-CLIFFS IRON COMPANY
MINING DEPARTMENT.
SAFETY INSPECTOR'S REPORT.

.......................Mine. Date.......................

(Sections and Rules quoted in this report refer to the Proposed

Uniform Mining Laws*). Answers must be made to the following as to the safety and health of the employes and suggestions must also be made for any improvement or betterment of conditions.

SURFACE—

1. What are conditions of roadways on Company's property over which workmen are obliged to travel to reach place of work?
2. Are such roadways properly lighted?
3. Are railway tracks used as roadways and cannot other ways be provided?
4. Are all open pits, caves, disused and abandoned shafts properly protected? (Rules 23-24.)

CHANGE HOUSE—

1. Is dry lighted by kerosene lamps or electric lights?
2. Is dry sufficiently lighted?
3. Are accommodations ample for men employed at the mine?
4. What protection for safety from fire is provided?
5. Is fire protection kept in good order?
6. What are the sanitary conditions?
7. What accommodations and appliances are provided for injured persons? (Sec. 21.)
8. How many fire-fighting helmets are provided? (Rule 6)—and where kept?
9. Are the same kept in good order?
10. Are there men around the mine who understand the first care of injured men and the use of fire-fighting helmets?

SHAFTS—

1. Is protection at collar of shaft sufficient and in good order? (Rule 51.)
2. Is opening to shaft at timber tunnel properly protected and in good order?
3. Is protection at shaft stations sufficient and in good order? (Rule 50.)
4. Are there skip tenders and at what levels? (Rule 7.)
5. Is there a cage rider?
6. What tools are allowed with men riding in skip, cage or bucket? (Rule 8.)
7. Are projecting tools properly lashed to hoisting ropes? (Rule 9.)
8. How often are timbers in manways cleaned? (Rule 22.)
9. Are they in good condition?
10. Are working stations sufficiently lighted? (Sec. 26.)
11. Are there passageways at all levels around hoisting compartment? (Rule 44.)

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*American Mining Congress, Vol. XIII, No. 9. Published at 1510 Court Place, Denver, Colorado.

12. Are same in good condition?
13. Are there guides for the bucket?
14. What style of crosshead is used? (Rule 15.)
15. What clearance is provided on guides?
16. Is stopper securely fastened to hoisting rope at least seven feet above rim of bucket? (Rule 15.)
17. Are guides and crosshead kept in good condition?
18. Is there more than one outlet to surface?
19. Are there connections between levels other than the main shaft? (Sec. 38.)
20. Is the second outlet kept in good condition? (Sec. 38.)
21. Is condition of pipes, electric wires and conduits safe?
22. Are steam pipes covered or protected from accidental contact?
23. Is the general condition of the compartment through which men are hoisted safe for men as regards lagging, timber, guides, etc?

SINKING—

(When more than one shaft is being sunk, separate report must be made for each). The following questions apply to sinking new shafts and also deepening shafts:

1. In lowering, is bucket stopped 15 feet above the bottom of shaft? (Rule 10.)
2. Is shaft suitably covered while sinking is going on? (Rules 11-12-13.)
3. Are only safety hooks used for bucket? (Rule 14.)
4. Are there ladders reaching to the bottom of shaft while sinking? (Rule 42.)
5. State what kind of ladders and if in good condition?
6. Is bell rope within reach of men in bucket at bottom of shaft?

GENERAL MINING—

1. Is general condition of timbering or other means of support throughout the mine satisfactory? (Sec. 12.)
2. Are mine maps clear and accurate for purpose of inspection? (Sec. 22.)
3. Are all dangerous places fenced off? (Rule 4.)
4. Are proper danger signal boards displayed at all dangerous places? (Rule 4.)
5. Are candles or lamps left burning after shift? (Rule 5.)
6. Are sumps securely planked over? (Rule 45.)
7. In passageways, are roofs and walls securely lagged? (Rule 46.)
8. Are winzes and raises in direct line of drift? (Rule 47.)
9. Are winzes, raises and open stopes properly guarded? (Rule 48.)
10. Are ladderways in winzes and raises located in drifts properly protected by hatches? (Rule 49.)

11. Are all chutes in winzes and raises properly protected by gratings?
12. Is proper provision made for safety of men working at chutes?
13. Are the communications between contiguous mines in good condition? (Sec. 39.)
14. What is the condition of ventilation in different parts of the mine? (Sec. 43.)
15. Are sufficient dry closets maintained? (Sec. 44.)
16. While the responsibility for the safety of the roof and walls in the individual places is upon the workman, are the same also inspected by superintendent, captain and shift bosses, and how often? (Sec. 45.)
17. Are the rules for safety pillars on boundaries observed? (Sec. 46.)
18. In approaching workings known to be flooded are proper precautions taken? (Sec. 37, Rules 32-33-34.)
19. Complaints from workmen—report if any?

MAIN AND SUB-LEVELS—

1. Are levels and sub-levels properly ditched to prevent water from accumulating, making passage unsafe?
2. Are there any other conditions which might be dangerous for passageway for men?
3. Are tracks and switches is safe condition?
4. Is condition of wiring safe?
5. Is condition and arrangement of piping safe?
6. On haulage roads, are drifts properly lighted? (Rule 26.)
7. On all main drifts is there a manhole or refuge every 100 yards? (Rule 28.)
8. Are they kept clear?
9. Are the rules governing the employment of motor and brakemen carefully enforced?
10. Are the rules governing the operation of electric motors strictly enforced?

EXPLOSIVES—MANNER OF HANDLING—

1. Are explosives stored on surface and is Section 25 strictly complied with?
2. Are caps stored on surface, and where?
3. How often are explosives taken into mine?
4. Are packages of explosives plainly marked, giving date and strength, also name and place of manufacture? (Sec. 26.)
5. Is there more than 72 hours supply of explosives stored in one place in the mine? (Sec. 25.)
6. Is the same stored 50 feet from any other supply of powder? (Sec. 25.)
7. How is dynamite thawed? (Rule 57.)
8. Is the method employed safe? (Rule 57.)

9. Are conditions as to blasting complied with? (Sec. 30, Rules 56 to 65.)
10. Are the rules posted in conspicuous places?
11. Do men in preparing dynamite for blasting always use skewers and cap crimpers?
12. Before blasting are the necessary warnings given?

LADDERWAYS—

1. Do rungs of ladders exceed 12 inches apart? (Rule 35.)
2. Are ladders placed 3 inches from wall of shaft or other opening? (Rule 36.)
3. Are there platforms for ladders at least every 24 feet? (Rule 37.)
4. Are there manholes for ladders? (Rule 38.)
5. Do ladders project at least 3 feet above sollars, or are hand rails provided? (Rule 39.)
6. Are any ladders inclined backwards from the vertical?
7. Are rungs made of wood or iron?
8. Is there a complete ladderway from the lowest workings to surface? (Sec. 42.)
9. Is there any loose rock or timber left at the top of ladderways, where it might accidentally fall or be pushed into same?
10. Are ladders and ladderways kept in good condition?
11. How often inspected?
12. Are stairways kept in good condition?

HOISTING—

1. Have printed rules, Sections 31 and 32, governing duties of hoisting been furnished to hoisting engineers and receipt taken for same?
2. Are Sections 31 and 32 governing duties of hoisting engineers properly posted? (Sec. 32, Rule 14.)
3. Are they maintained in good condition?
4. Is speed of hoisting and lowering men over 1000 feet per minute? (Sec. 32, Rule 8.)
5. How often are machinery and safety appliances inspected? (Sec. 32, Rule 9.)
6. Are cages, skips or buckets used for hoisting men, inspected daily? (Sec. 34.)
7. Are the cages properly enclosed and protected?
8. Are signals in good order, is there a duplicate system of signals? (Sec. 37, Rules 16-17.)
9. Is signal code posted in engine house and at each level? (Sec. 37, Rule 21.)
10. Is moving machinery properly guarded? (Sec. 35.)
11. Is hoisting plant equipped with an overwinding device?
12. Is the same in good working order?
13. Are safety devices tested once each month by the superintendent and inspector jointly?

14. Are hoisting ropes running through engine house protected for safety of men?
15. Are unnecessary persons excluded from the engine and boiler houses? (Sec. 32, Rule 5.)

HOISTING ROPES—

1. Are ropes inspected every 24 hours? (Sec. 33.)
2. What is the present condition of rope?
3. How often is section of rope tested? (Sec. 33.)
4. When rope is lowered to bottom, are there two full turns left on drum? (Sec. 37, Rule 53.)
5. Is the further end of rope secured by six clamps or bolts? (Sec. 37.)
6. Is rope treated with suitable oil or compound monthly? (Sec. 37, Rule 55.)
7. How is lubricant applied?

SURFACE—

1. Are sheave stands and sheaves in shaft houses easily accessible and arranged for the safety of the men whose duties require them to oil and repair the same?
2. Is the machinery and belting in shops properly protected for the safety of employes and others?
3. Is the fire pump hydrant equipment provided for fire protection, ample?
4. Is the quantity of fire hose provided ample?
5. When last inspected?
6. What is condition of same?
7. Where are lubricating oils and greases stored?
8. Is storage place kept in safe condition?
9. Where are kerosene or inflammable oils stored?
10. Is storage place kept in safe condition?
11. Are hoisting ropes, guy wires and piping on surface properly protected?
12. Is protection provided for safety of employes and others during construction and repair work?

BOILERS—

1. Are safety devices provided for boilers, kept in good working condition?
2. Are general conditions in boiler house safe?
3. How often are boilers inspected by insurance companies?
4. Are their recommendations promptly complied with?
5. Are all steam pipes covered or protected from accidental contact?
6. Is boiler room properly lighted?

ENGINE HOUSE—

1. What is condition of piping and general equipment as regards safety?
2. Is engine room properly lighted?

3. Is basement of engine house used for storage of material?
4. Is the same arranged for safety to men?

COMPRESSORS—

1. Is moving machinery properly guarded? (Sec. 35.)
2. What is condition of piping around compressor as regards safety?

ELECTRIC ENGINES AND DYNAMOS—

1. Is moving machinery properly guarded? (Sec. 35.)
2. Is switchboard railed off to prevent handling by outsiders?
3. Are all wires carefully strung to prevent accidental contact?

PUMP STATIONS—

1. Is moving machinery properly protected?
2. Are electric wires safe?
3. Is station properly lighted?
4. Are steam pipes properly protected from accidental contact?

TOP TRAM—

1. Is moving machinery properly protected?
2. Is wiring properly protected?
3. Are counterweights safe?
4. Are walks and hand rails provided on permanent trestles?
5. Can the conditions for safety of men be improved?

CRUSHER—

1. Are belts and shafting properly protected?
2. Are proper railings provided for all walks and openings?
3. Are electric wires properly protected?
4. Are stairs and ladders in safe condition?
5. Are all oiling places safely accessible?

STEAM SHOVEL—

1. Is all moving machinery properly protected?
2. Are all gears covered?
3. Is shovel in good condition for the work being done?
4. Are runways and stands protected by railing?
5. Is proper care taken to prevent trestle legs from falling and injuring workmen?

GENERAL APPLICATIONS—

1. Are accidents to persons and equipment reported promptly?
2. Are occurrences of fires, floods, extraordinary caves, and similar accidents, dangerous to life or health, reported promptly? (Sec. 12-16.)
3. Do superintendents, captains and shift bosses, instruct employes as to their responsibility?
4. Is discipline strict or lax?
5. Are persons allowed in or around mine except by permission? (Rules 66 to 70.)

PAPERS PUBLISHED BY THE INSTITUTE, WHICH TREAT OF MINE SAFETY AND SOCIAL CONDITIONS.

DISCUSSION.

MR. HEARDING: I wish to ask if, for instance after an accident occurs, is there any investigation made by the safety committee of the accident, its origin, the causes that led up to it, and if possible a clear outline of those causes with the idea of correction.

MR. CONIBEAR: I would say that the company has an accident committee, consisting of three superintendents; this committee investigates all fatal accidents and makes out a report to the agent of the company. In addition to this, we have a central safety committee which receives reports of all accidents, at which time the causes of these accidents are duly considered. The superintendents of the mines, at the time accidents occur, are there to get all the information; in addition, all accidents that are very serious are inspected by the safety inspector, who can corroborate their statements.

MR. DUNCAN: This central safety committee passes on the causes of all accidents and their prevention. If we find that these accidents are preventable, then we add to the rules already in force, such rules as are necessary to prevent the recurrence of that particular accident. In that way we have added several rules to those that were originally adopted.

MR. KELLY: I would like to ask if those rules are incorporated in this paper?

MR. CONIBEAR: No, the rules are not incorporated in the paper.

MR. DUNCAN: I would like to say, on behalf of our company that we will be glad to furnish any member of the Institute requesting them, the rules governing our safety inspection system. They are published in two forms, one for the superintendents and mining captains and the other for the workingmen, and copies of these rules are given to the workingmen who sign a receipt for them and they are expected to so far commit them to memory that they can practically pass an examination upon them. Those refer particularly to the operating conditions underground. Those to the superintendents are more voluminous, containing all the rules that they are required to see are enforced.

MR. DENTON: We take it for granted, of course, that these gentlemen are satisfied with the progress they are making in this direction. Can they tell us about indications of results that are coming?

MR. CONIBEAR: I feel very confident that the workingmen realize now that the system of safety inspection has reached a practical stage. The discipline of the men, in my opinion, is very much better. Of course we are all willing to acknowledge that accidents can be reduced, but the system has been in existence such a very short time that this is a subject that we do not care to discuss.

MR. DUNCAN: You might indicate the proportion of reduction in accidents.

MR. CONIBEAR: We find that there is a reduction in minor accidents of 16 to 20 per cent.

MR. SMYTH: It seems to me that the rules should be published as an appendix to Mr. Conibear's paper.

MR. DENTON: I would say, as a bit of information for this Institute, that in line with Mr. Kelly's resolution which was introduced a year ago and adopted, authorizing the Council to appoint committees on various subjects, a committee with the title "Committee on the Prevention of Accidents" has been appointed, and it is understood that one of the main duties of that committee will be to collect just such information as will be represented by these rules, and publish it for the benefit of the members.

MR. HEARDING: I think it would make a rather voluminous addition to the Proceedings unless they are unusually brief. I happen to be interested in printing some rules myself and I think they cover about three hundred pages.

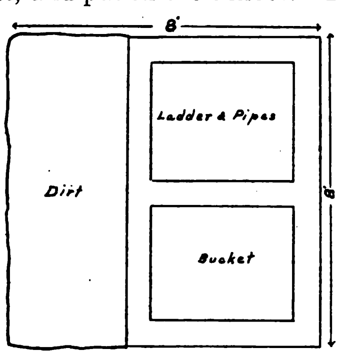
MR. DENTON: I feel sure they would be a very valuable contribution to the proceedings of the Institute; the rules themselves would convey a large amount of information. I think this is just a beginning and that during the next year or two of the Institute, we may be able to collect a great deal of additional information along this line.

RAISING SHAFT AT ROLLING MILL MINE, NEGAUNEE, MICH.

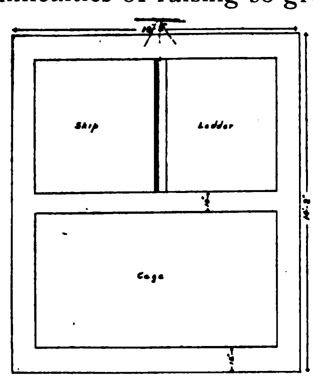
BY EDWIN N. CORY, NEGAUNEE, MICH.

The new shaft of the Rolling Mill Mine of the Jones & Laughlin Steel Co., was raised from the 621-foot level to the surface by the following method:

The work was started from this level and carried through to surface with one continuous raise. A force of nine miners divided into three shifts of three men on each shift working eight hours per day was employed. These men also trammed the material from the raise to No. 1 shaft, a distance of 750 feet, and put in the timber. The difficulties of raising so great



Plan of Raise.



Plan of Shaft

a distance were successfully overcome and no accidents or delays occurred during the progress of the work.

The raise was 8x8 feet in size and was divided into three compartments. One compartment, 4x4 feet, including the timber, was used as a bucket way for hoisting tools and timber and a station was cut at the bottom of this compartment on the main level, in which was placed a small hoist. Another

compartment, 4x4 feet, was used for ladders and an 8-inch pipe from air fan, also one 1¼-inch pipe for speaking tube and one 1½-inch air pipe for the power drills. The other half of the raise, 4x8 feet, was used for the rock broken in the raise, and was not timbered but kept filled with rock up to the height of the timber in the other compartments. A chute was constructed in the bottom of this compartment through which the rock was run when loading into tram cars. (See Fig. 1.) A fan was placed at No. 1 shaft which forced the air through the 8 inch pipe up to the top of the raise, the current going down through one of the compartments, thus securing perfect ventilation.

The type drill used was an Ingersoll-Rand, 2-inch cylinder, B. C. 21, butterfly-valve hammer drill. Three drills were operated at one time to drill the raise over, which consisted of 18 holes in three rows, six holes in each row. The cutting-in holes were drilled, so as to cut a space when blasted through the entire length of the raise directly over the rock compartment (Fig. 1) so that the other holes would throw the rock toward this opening and fall into the rock compartment. Before blasting, the ladder and bucket compartments were covered with 10-inch round timber, flattened on two sides to prevent rolling, and placed at an angle to deflect the falling rock into the rock compartment, necessitating only a minimum amount of shoveling. The space directly over the ladder compartment, was covered with 3-inch plank to permit the men to get away quickly when blasting, also to give them perfect protection in going up and down the raise (Fig 2). The blasting was all done with fuse cut in various lengths to give the desired results. After blasting, and while the smoke was clearing out of the raise, the miners would tram enough rock out of the chute so that timbering could be commenced at the top of the raise. A gin-pole was erected about 8 feet from the last set of timber on which was hung a 10-inch sheave wheel for the rope used in hoisting the timber from the level below (Fig. 2). After the timber had been put in it was also

used for hoisting the sharp drills and machines to start drilling. The drilling machines were thoroughly overhauled after each round of holes while the miners were blasting and timbering, and no delay was occasioned during the work.

When the raise had been carried to a height of about 200 feet, hitches were cut in the rock and a set of bearing pieces

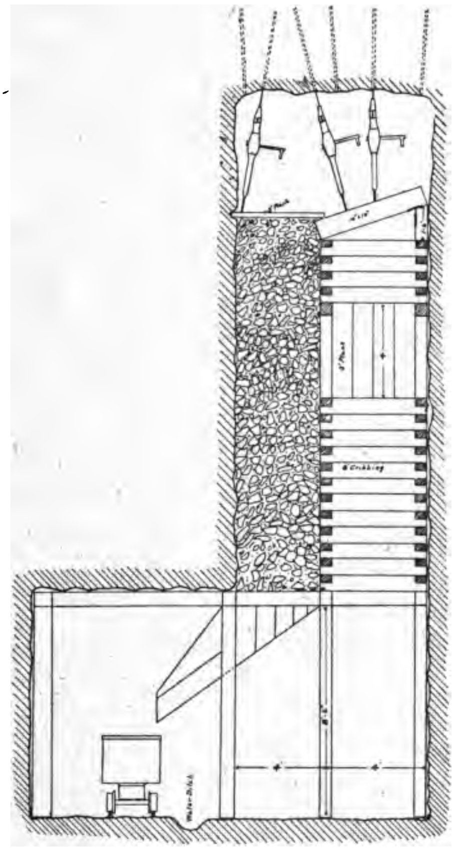


Fig. 1

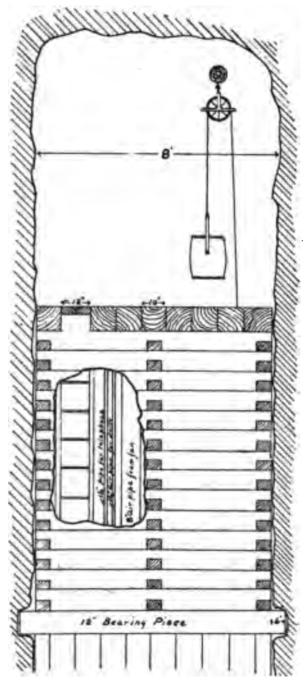


Fig. 2

put in about four feet above the last set of timber, and planked up with 3-inch plank. This was done to take the weight of the timber for the next 200 feet instead of letting it rest upon the timbers below. (See Figs. 1 and 2). A station 15 feet long was cut in the side of the raise about every 200 feet, to shel-

ter the men when blasting instead of going down the ladders the whole distance to the level below.

In this manner the raise was carried to the height of 570 feet, which was 51 feet from the surface, when a smaller raise was carried up as is shown in Fig. 3, a test hole being drilled in advance of the top of the raise to ascertain the depth of the sand. When within 18 feet from the surface sand was

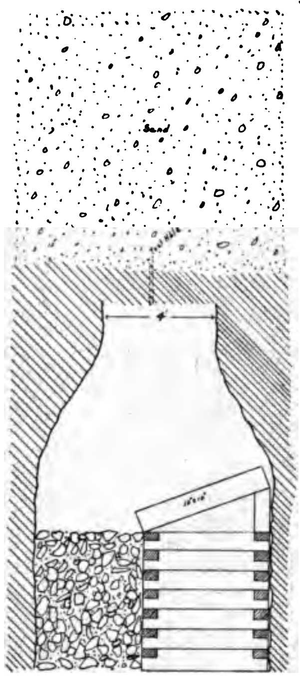


Fig. 3

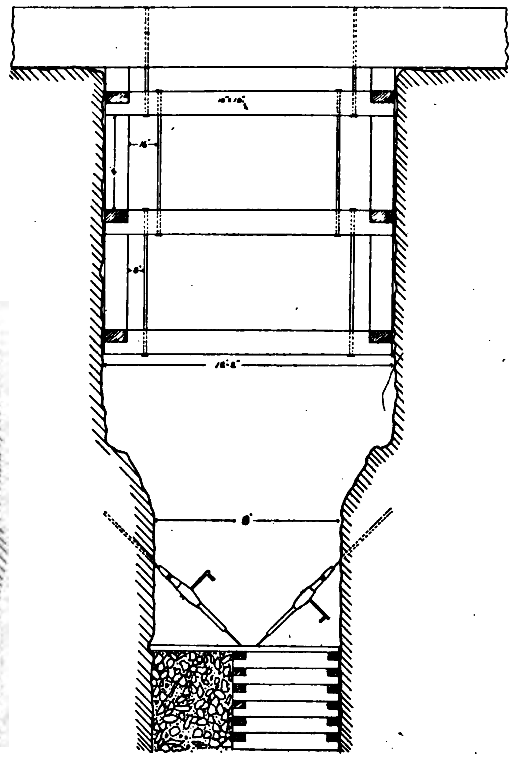


Fig 4

reached by the test hole, and the raise continued, by carefully working through the sand to within 10 feet of the surface A hole was then drilled from the surface, blasting the sand through to the opening below.

The work of enlarging the shaft was then begun. The surface at the opening made by the blast was leveled off and

two stringers, 2 feet 6 inches in diameter and 25 feet long, were placed in position for timbering through the sand. The dimensions of the shaft are 10 feet 2 inches by 12 feet 2 inches within timbers; 12x12 inch fir timber was used and the plan of timbering is shown in Fig. 4. The principal feature of this work was the method used in enlarging the opening made by the raise to the dimensions required by the permanent shaft. The type of hammer drills used in raising was also used for this work. The rock was drawn off through the chute at the 621-foot level as in raising until it was lowered about 15 feet. The timber from the raise was then pulled out for a distance of 15 feet and the two smaller compartments covered over as before. The holes were drilled upwards as in raising, but at an angle of 45 degrees, and were started 10 feet below the permanent shaft timber and so located as to strip 5 feet of the shaft at one blast, thus making room for one set of the shaft timber.

Ordinarily this work is done by drilling holes downward, but, as the above method proved very successful, it was continued for the entire shaft, and I think the use of the hammer drill a great improvement over the reciprocating type with shaft bars or tripods.

The progress made in the work was as follows:

Raising—621 feet; No. days, 125; average per day, 5 feet.

Cutting down—621 feet; No. days, 114; average per day, 5.44 feet.

Work started September 5th, 1911; completed July 15th, 1912.

MINE SANITATION.

BY E. B. WILSON, SCRANTON, PA.

An examination of the list of papers printed by the Institute will show that three papers dealing with sanitation have been presented. While there is little new in this paper it is hoped that it will have the effect of concentrating the thoughts of all on this important subject.

Owing to various circumstances mine workers are not so plentiful as heretofore and the mining industry is confronted with such economic conditions that it must care for the miners it has and offer sufficient inducements to entice men to become miners. Aside from this there has been a wave of generosity sweeping the country for the past few years and both miners and mine laborers are treated with vastly more consideration and recognized as an integral part of the industry. Underground sanitation of mines is of particular importance at this time because of the dreaded hook-worm having obtained an entrance into mines in this country.

Dr. B. W. Jones* in his paper on the Importance of the Ordinary Prevention of Water-Borne Diseases in Mines, showed that typhoid fever, tape worm and hook-worm could be contracted by drinking impure water in mines. Germs of disease and the infectious parasites are transmitted from one person to another in many ways but chiefly through the air and through water. In every metal mine employing over 25 men arrangements should be made for evacuation. Latrines should be provided but if they are not kept clean the conditions may become almost as bad as if there were none. Sheet iron cans or boxes should be arranged for receivers and into

*Lake Superior Mining Institute Proceedings, Vol. XII, p. 105.

these dry sawdust and ashes should be sprinkled to act as deoderizers. Chloride of lime is not considered a suitable deodorizer for underground. These cans and boxes should be provided with covers; taken to the surface; cleaned and returned daily. The roof and sides of the places should be kept fresh with white wash; the floor cemented and sloped so

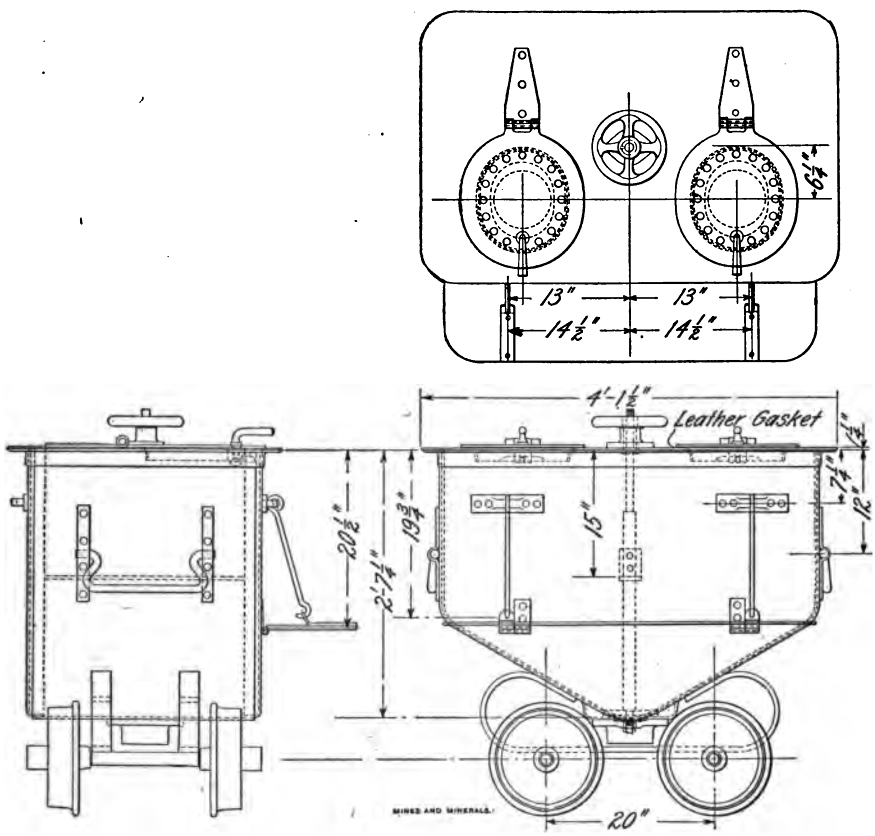


Fig. 1. Anaconda Toilet Car

that it may be washed and not collect any filth. For privacy portable screens might be arranged.

In some of our large mines special arrangements are made for the relief of men during working hours and besides underground latrines similar arrangements are provided near

the shaft mouth. Outside arrangements should not be near the intake airway and underground arrangements should be in the return airway.

The following description of a toilet car used in Montana was furnished by A. W. Charles:

This toilet car from a sanitary point of view is a great advance over the old system which prevails at most ore mines.

The detailed drawing, Fig. 1, gives the dimensions of this car, but a few words in explanation of its construction may not come amiss. The car body which is of 3-16-inch steel plate is mounted on ordinary mine-car wheels. The top to the car is portable and is secured by ½-inch snap-headed bolts. Leather gaskets are placed between the body sides and the top, as ordinary rubber gaskets become too soft after the car has been in use some time.

The seat covers are hinged, and supplied with leather gaskets which are secured to the covers. The covers are provided with handles that when turned latch them so securely to the top of the car that it may be hoisted up the shaft without any disagreeable results. There is a portable hand wheel on top of the car which, on being revolved to the right or left, closes or opens a discharge valve in the bottom of the car. The stem connecting the valve with the hand wheel extends through the car as shown in the drawing. The car is discharged over a vault and cleaned out with a hose.

Water and a little chloride of lime are placed in the car before it is lowered into the mine. The cars which are being used quite extensively at the Anaconda mines, could be adopted to great advantage at other mines where sanitary conditions are lacking.

The hook-worm is present in the Cornish tin mines in the Camborne District, England, and has been for some years. It is becoming almost general in the southern coal mines of the United States and it is almost sure to enter our deep northern ore mines unless each operator uses great care to prevent it. Mines which become infected require the most drastic and expensive measures for universal extermination of the pest; in fact complete eradication does not yet appear to have ever been accomplished. According to the report of the Committee of the British Association for the Advancement of Science*, "The necessary prevention is best accomplished by examination and limited quarantine of new comers to work in the mines; provision of proper sanitary accommoda-

***August, 1904, page, 292, Institution of Mining Engineers, Vol. 28, p.707.**

Fig. 2. Change House at the Marianna, Pa., Mine, of the Pittsburg-Buffalo Coal Co.

tions underground and at the mine mouth; and regulations to prevent pollution of the mine." In some mines it is customary for the miners to place an empty can or bucket where it will fill with water dropping from some crack in the rocks or running from above along the walls. It will be necessary to prohibit this and furnish pure drinking water from the surface keeping it covered in proper vessels; or as in some cases sending the water down into the mines from the surface in pipes. This is economical where there are many men at work on several levels.

Dr. Augustus Koenig** mentioned three kinds of Fibrosis that miners were subject to, namely: Anathracosis, Calcicosis and Siderosis. The most important one Silicosis was omitted. This leads to consumption with which quartz miners are afflicted particularly in South African gold mines. Sometimes the healthiest men will last only six months and at the end of the year phthisis accounts for 300 of every 1,000 death among natives alone.

Silicosis is due to fine particles of silica dust getting into the lungs and irritating them causing fibrosis; thus affording the germs of consumption an opportunity to lodge, increase and destroy the lungs. Silicosis is not infectious, the infecting agent being bacillus conveyed through the sputum of infected persons.

Consumption or tuberculosis due to silicosis is not so prevalent in the metal mines of the United States as in Cornwall, England, Bendigo, Queensland, West Australia and the Transvaal. The death rate per thousand from miners' phthisis is as follows:

Cornwall, England	5.3 per 1000
Bendigo, Victoria	8.8 per 1000
Queensland	2.54 per 1000
West Australia	3.42 per 1000
Transvaal (White)	8.48 per 1000
Transvaal (Natives)	23.10 per 1000

**Lake Superior Mining Institute Proceedings, Vol. XII, p. 113.

The Royal Commissions of those countries and New Zealand have recommended the following as a preventative to this disease:

"The compulsory use of dust preventing appliances such as sprays, water blasts and atomizers.

Improved ventilation of mines.

Use of bath and change houses at mines.

Prevention of indiscriminate spitting and the destruction of tuberculosis sputum.

Definite treatment of those affected with tuberculosis of the lungs in an advanced form.

Improved housing conditions and disinfection of work places and living quarters.

The exclusion from work underground of all persons infected with tuberculosis of the lungs."

So far we have never heard of a metal miner in the east having silicosis and very few anthracite miners have anthracosis even in the anthracite fields where the coal is sharp.

Coal mines are very dusty at times but the coal miners are able to clear out their throats and lungs. Lead ore miners are not so troubled with silicosis as from being leaded by the dust. In such mines there should be water sprays at all machine drill holes and the ore wetted to keep down the dust when shoveling and dumping. Ferricosis and cupricosis must be rare in Michigan as no one seems to have heard of these diseases, but that is no reason why the ventilation of the iron and copper mines should be neglected, for as a preventative to disease pure air stands forth pre-eminent.

In 1886 the writer had charge of an iron mine in the east which was wet. The men came from this mine quite wet and it was nothing short of crime to permit them to walk home in wet clothes on cold days. A change room was therefore rigged up where the men could also wash and put on dry clothes. It was part of the boiler house and soon became so infested with what the men termed "grey back" (and judging from the description given these things were not of the

"snipe" genus), that it was given a wide berth. The men used buckets for wash bowls and spillaged (to use an English expression) the water over the floor, thus keeping it in a soggy condition. There was but one window in the room, consequently it was always dark. Water was heated from the pump exhaust and sometimes there was no cold water and vice versa. The room had one redeeming feature, it was part of the boiler room and was never cold summer or winter. After reading Mr. William Kelly's* paper on the Change

Fig. 3. Change House of the Universal Coal Co. in Indiana

House at West Vulcan mine the comparison seems to be on a par with the foot tub and bath tub. Critically examining the West Vulcan change house we find there are double lockers for each man, one for dry clothes and one for shift clothes. The water basins are portable and there are no shower baths.

The Change House at the Cliffs Shaft mine, described by Mr. J. S. Mennie**, reduces the lockers to one per man; has

*Lake Superior Mining Institute Proceedings, Vol. VIII, p. 70.
**Lake Superior Mining Institute Proceedings, Vol. IX, p. 121.

central drying frames for men's shift clothes; has shower baths and wash room with portable basins. If a wash house is fitted with wooden lockers there will be difficulty experienced in keeping down vermin; in fact the less wood there is about a wash house the better. Hydrocyanic acid will kill the eggs of anything; it is however so deadly a poison it can only be used with extreme caution and by a skilled man. Formaldehyde has been recommended but is not effective; is expensive and leaves a penetrating odor in the clothing. Carbon disulphide, a heavy liquid, boils at 115 degrees F. and rapidly saturates the surrounding air with its vapor. This liquid if applied at the top of wooden lockers will vaporize and fall through the locker, penetrating the cracks on its descent and kill any vermin. It does not kill the eggs however so its application is more in the nature of a fumigant which is to be repeated as often as the exigencies of the case demand.

The latest in change houses is that at the Marianna, Pa., mine of the Pittsburg-Buffalo Coal Company, an interior view of it being shown in Fig. 2. Each miner has his own hanger, lifts his clothes up to the ceiling and locks the chain to the steel rack. The clothes are then out of reach from the floor and are exposed to light and ventilation. This arrangement is superior to lockers in every way and for drying out damp clothes it is excellent. There are 90 shower baths in this house, also a few bath tubs and stationary wash basins.

In Fig. 3 is shown the Universal Coal Company bath house in Indiana with the clothes hangers, wooden racks that are also benches, and wash bowls arranged along the side of the building. It should be demanded of the men that they put on clean working clothes at least once each week, and that the dirty clothes be washed, dried and aired. At some mines the men hire the man who cares for the bath house and sees to the drying of clothes. This arrangement has the advantage that the men cannot blame the company if the house is not kept in order.

The constant exposure of men to warm temperature in more or less vitiated air underground is apt to thin their blood

and render sudden changes in wet clothing to colder conditions above ground injurious to health. This is amply shown by the tables arranged by the Transvaal Chamber of Mines where tropical natives died from pneumonia at the rate of 36.43 per 1000 per annum during 1911. To overcome this the natives are given blankets as they come from the mine, also hot coffee as soon as they reach the change house where they are compelled to stay 20 minutes in a 60 degree F. temperature; and are then given more hot coffee when they leave the change house.

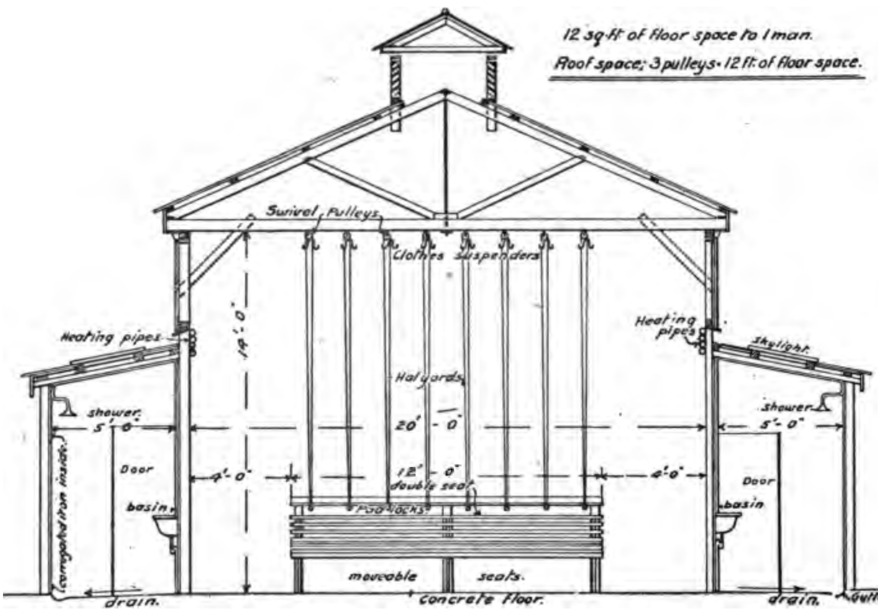


Fig. 4. Design for Change House as found in the Report of the Royal Commission of New Zealand for 1911.

In looking up data on change houses the design shown in Figs. 4 and 5 was found in the Report of the Royal Commission of New Zealand for 1911. It has a capacity for 50 men on one shift and allows 12 square feet of space to one man but three pulleys are in this space, consequently it accommodates 150 men in three shifts. The hook-and-pulley system is the best design from a hygienic standpoint for reasons noted. In cases it may be advisable to furnish metal wash tubs that men may wash their mine clothes. It is probable that one basin and one shower bath will serve for

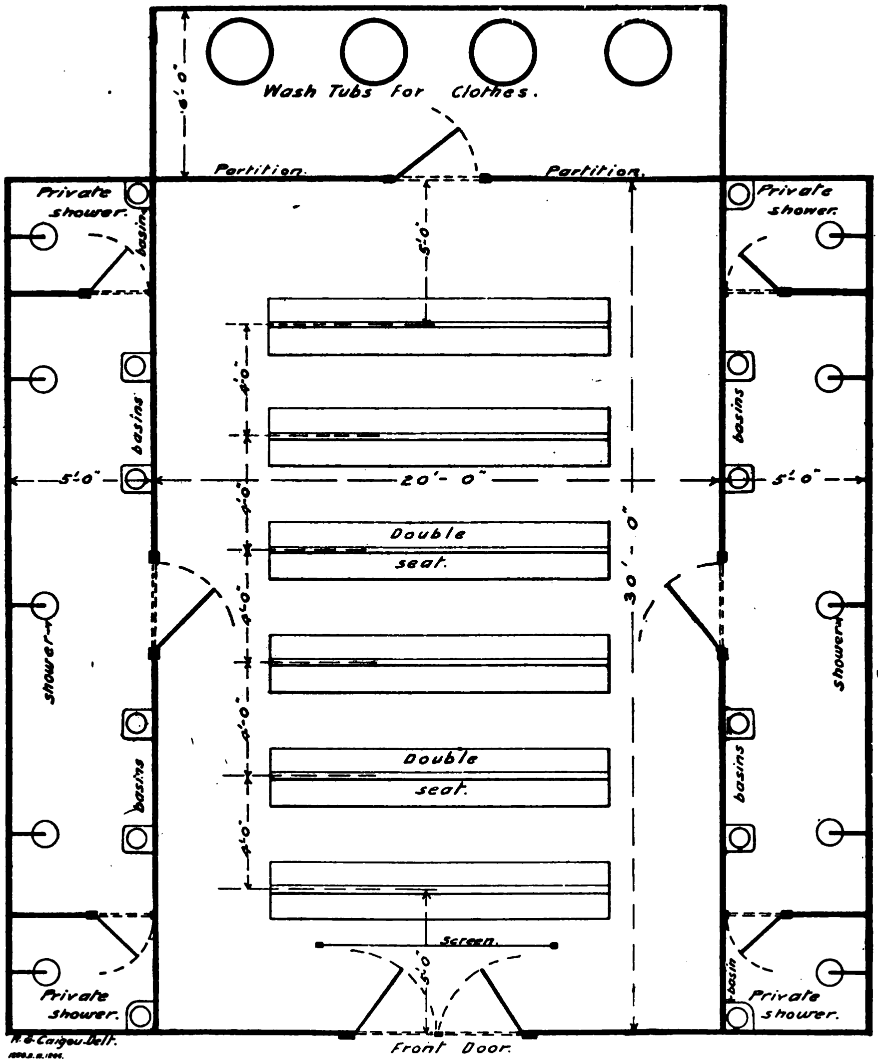


Fig. 5. Design for Change House as found in the Report of the Royal Commission of New Zealand for 1911.

every five men employed underground. Concrete floors with proper drainage are requisite in such houses and stationary wash stands and one or two bath tubs are advisable. Rules for the keeping of the wash rooms and change rooms clean are imperative if they are to be sanitary.

UNEXPLORED PARTS OF THE COPPER RANGE OF KEWEENAW POINT.

BY ALFRED C. LANE, TUFTS COLLEGE, MASS.

In one of the recent magazines†, there has been a plausible criticism of certain geological surveys for coming after the mining operation and not guiding it. There has also been a statement recently made‡ that the "copper district has been pretty thoroughly explored." It therefore seems timely for me to point out some parts of the district not yet explored, where property seems as good as in the regions where the paying mines are. For my report on the Keweenawan series (Publication 6), which finishes my work as State Geologist, is just out, and there is also the report on the geology of the Lake Superior region, issued by the United States Geological Survey, which contains a long and valuable chapter on the Keweenawan series§.

My report is already a back number. The complaint so often made of such reports may well be made of it. It is practically of date 1909. My embarrassment is lessened by the fact that I can also announce that Publication 8 of the Michigan Geological Survey on the Mineral Resources of Michigan, by my able successor, Dr. Allen, is also practically ready and brings the tale down to 1911. I understand they will do some work also this year.

One reason why geologists have been loath to guide exploration is that it is almost never possible to say where exploration and mining will surely be a commercial success,

†Mining World, p. 1138, 1912.
‡Appraisal of Mining Properties of Michigan by the State Board of Tax Commissioners, 1911.
§Monograph 52, by Van Hise and Leith, Chapter XV.

while it is entirely possible to see where it will not be a commercial success. Thus, while one from my report* might well gather, what most of you well know, that it is relatively useless to explore in heavy beds of trap like the Greenstone, or heavy beds of conglomerate like the Great Conglomerate, and that very heavily shattered zones are likely to be poor, he can not so well gather just where to look. In a general way, the most favorable place will be in a belt near a heavy impervious belt, and in a somewhat open pervious one, in which a certain amount of shattering has taken place, but not too much, and at a depth where the water and rock are already somewhat salty.

But just where are such places? One's reputation as a prophet is much safer if one turns down a proposition than if he recommends it. For the commonest kind of proposition is where the chances are something like ten to one against getting a return of one hundred to one on the investment. That is well worth trying, though it will, in nine cases out of ten, lead the geologist to be blamed, and in the tenth case of success, many others will deservedly receive a large portion of the credit.

At the same time, geology has, in many cases, helped the miners. I have only to call attention to the work on the Mesabi Range, described by H. V. Winchell, some years ago, for your Institute, and the work of Dr. L. L Hubbard and others of your members, and I am not afraid of hurting my reputation by pointing out that the Copper Range is, by no means, as yet "pretty thoroughly explored," in places where chances are as favorable as they were in advance for any of our successful mines.

We are now all pretty well agreed that the copper was deposited by water and that these waters were circulating downward and were hot and that at least the original deposition of the copper took place, therefore, during, or shortly after, the Keweenawan period, while the original volcanic

*pp. 41, 487, 497, 799, 813, 850, 875, 885, 945.

heat was not yet dissipated, as it is so thoroughly today. We find, therefore, the copper deposited in and near the beds originally pervious, and, generally speaking, we find the hanging wall, if these beds are not absolutely vertical, an impervious bed.

The first question we may ask ourselves is whether all such porous and pervious beds have been explored. I need hardly say that this is far from the case. Leaving out of account for the moment the great porous sandstones and conglomerates at the top of the formation, which contain at least one commercial lode—the Nonesuch lode—we have twenty-two smaller conglomerates which Marvine counted at Portage Lake, and there are several of minor importance which he did not reckon, and we also have a large number of amygdaloids—something over two hundred and eighty† which were originally porous and, as is well known, certain amygdaloids are copper bearing.

Now, how many of these have been developed by mining to any extent? Mining has been confined to about six horizons in the amygdaloid—the horizon of the Ashbed and Atlantic, that of the Pewabic, Quincy and Franklin, Jr., that of the Osceola amygdaloid, that of the Kearsarge amygdaloid, that of the Isle Royale lode and Winona, and that of the Baltic lode. Some of these horizons include several amygdaloids which are more or less tested up just around Calumet and the whole country between the Ashbed and Osceola lode or even the Kearsarge has been frequently cross-cut in many places by the system of shafts around Calumet. But do any such cross-cuts really show that the lodes crossed, though barren where cross-cut, may not somewhere else be valuable? Certainly not. It must be remembered that the Osceola lode on the Calumet property was cut many times before it was finally opened up and found by careful and experienced hand-

†Something less than 65 between the Great Conglomerate and the Pewabic West Conglomerate, about 172 between that and Conglomerate 8, and not less than 27 between Conglomerates 8 and 3, and certainly 15 or 20 below that.

ling that it might be profitably worked. It must be remembered that the Calumet lode (conglomerate) has been opened in many places other than the Calumet property and in no places outside the original region has been a commercial success. It must be remembered that the Allouez or Albany and Boston conglomerate, which has been opened at a number of points comes very near being a commercial lode, and would be such at present prices of copper and outside the Franklin Jr. mine has shown copper at other points, but, in most cases, not in paying quantities.

The fact, then, that a given amygdaloid has been traversed by some drill hole or even by some shaft and showed nothing to warrant special investigation among the other amygdaloids struck does not prove that it may not somewhere be a commercial lode. I do not consider it at all probable that commercial copper is confined to these six amygdaloids and the three sedimentary horizons in which it has been found. So that if these six horizons had been fully explored and the two or three conglomerates beside, and we suppose each to have incidentally involved investigation of three other lodes, there would be but an eighth of the range explored, unless we suppose that the accumulation of copper was confined to these beds.

The question arises whether these nine horizons above mentioned are likely to have been, in the first place, especially rich. It may be that they were formed in an interior basin, charged with salt volcanic waters and that these waters contained greater quantities of copper chloride at certain times than others, so that certain stratigraphic horizons are more likely to contain copper. The fact that the Kearsarge lode, for instance, has for nine miles, at least, been a commercial proposition and even where it has not been of commercial value has shown more than a normal amount of copper much farther, indicates to my mind some such possibilities. Yet there is no probability that copper is confined to these horizons, for we find it wandering across in fissures. I expect new

lodes and new horizons to be mined and it is unwise to hastily identify old lodes and new.

But assume for a moment that these nine horizons, which, even if we assume a maximum breadth of 200 feet each, are not 10 per cent of the formation, are a little more likely to contain copper than the average horizon. How far and under what conditions are they most likely to be workable and have they been "pretty thoroughly" developed under those conditions?

It is fair to assume, I think, that inasmuch as the copper is due to circulation, where the formation has been relatively undisturbed and the lodes are relatively flat, there will be less circulation and less accumulation of copper.

For instance, out on Keweenaw Point, where the dips are relatively low in the upper part of the formation it would seem to me that the bedded lodes would be less favorable to circulation and high concentration. On the other hand, cross fissures which are nearly vertical should be the lines of absorption, and hence the lines where copper would be in part accumulated. That seems to be fact. Bedded lodes out on Keweenaw Point hardly seem to run high enough in copper to pay, so far as explored. And yet, out on Keweenaw Point, the lower part of the formation below the Kearsarge, where the dips are steeper, has been but very little tested. Most of the work has been done in regions where the dip was less than 30°.

Once we go south of the Houghton tier (T. 54 N.) and hardly anything but a little diamond drilling and superficial trenching has been done above the horizon of the Isle Royale lode. While diamond drilling is a very valuable method of exploration and especially rich in its geological information, yet so far as the distribution of copper in a given lode is concerned, it is about like hunting for a needle in a hay stack with a magnetized steel rod. In the tier T. 53, the little that has been done at the Challenge and Globe covers only the lower or Bohemian Range formation and the work has

by no means shown absence of copper. Possibly if the copper market had been then what it is today, the work at the Challenge would still be going on.

That is one thing which must not be forgotten—namely, there is no hard and fast line between a commercial and a non-commercial proposition. What may be a commercial proposition one year may not in another year, and the explorations of fifty years ago can not be used, without careful consideration, in determining what is wise today.

I have mentioned the Central mine vein. According to all authorities these cross fissures affect the distribution of the copper in the bedded lodes. How much has been done in systematically tracing cross fissures? Very little, except so far as it can be done by geologists on the surface, and that is entirely insufficient, so far as prospecting for copper is concerned. The heavily drift covered areas, such as that south of Houghton, are untouched. The great cross valleys like those I mention may be recognized by the geologist, but have not been tested by the mining men. Diamond drilling has been almost entirely to get cross sections, and to test bedded lodes, not to test veins.

I would call attention to the fact that almost all the drilling has been at right angles to the strike of the formation. There are good reasons for this. More bedded lodes can be tested with a given amount of drilling and any deviation in the direction of the hole makes less error in interpreting the results. But I think that at times exploration along the strike and across cross fissures might be worth while, by drill holes deliberately pitched not at right angles to the strike. We know that at the Central mine, the amygdaloids were richer near the main cross vein. If a horizon proved promising, might it not be enriched near some such vein, or, if lean on one side, be better on the other? In places where exploration from the surface is not convenient, as in the cross valleys of Portage Lake, the Fire Steel, Flint Steel, Ontonagon, etc., holes oblique to the strike might locate cross fissures which have been of im-

portance in guiding the deposition of copper. One such region which has not been "thoroughly explored" lies close to us—towit, the bottom of Portage Lake!

In tier 52 more has been done (T. 52, R. 36), but even here little has been done to see what might be available in the region of the Kearsarge, Calumet, Pewabic and upper lodes. When we get down to the Ontonagon district, exploration has been confined mainly to a belt between the Kearsarge and Baltic lodes. It is worthy of note that wherever exploration has been persistent enough and deep enough, it has not been without substantial encouragement.

When we cross the Ontonagon, the Victoria mine and the Nonesuch exist to show that the deposition of copper did not stop, but very little has been done. There seems no reason why on the other side of the Porcupines in the fold matching that where the Nonesuch occurs, there may not be cross faults which shall have produced disturbances and similar accumulations of copper. The early exploration only touched the high spots and these we know to be poor.

This brings up the question why have certain regions been tested more than others. The reasons are many. Two are most important—a relatively thin cover of drift and accessibility. Another is almost or quite as important—an ownership which has the means and enterprise to develop. There are other factors, such as ease of transportation, nearness to developed properties, etc.

Leaving the economic reasons out of account, let us look at those especially geological—the covering of drift. Most of the mines have started and are where, as in the Lake and Baltic, a lode happened to outcrop relatively rich and with little or no surface covering. (See Leverett's map in Publication 7 of the Michigan Survey). This absence of surface covering is of very great importance in promoting discovery and exploitation, but there is no especial reason why the lode should be especially rich where the drift covering is thin—in fact, there are some reasons why they should be especially

lean, as such points might well have the copper leached out of them.

Again, according to my theories, the occurrence of copper is associated with the presence of salt water and we find in many cases, for the first few hundred feet, the salt water is gone. It seems altogether natural to suppose that the copper would be carried out or down, and there is some direct reason to believe this to be the case. In such cases, the appearance of lodes at the surface would be no sure sign that they might not contain richer copper at greater depths. The Calumet lode itself seems to have held copper rather more evenly a few hundred feet down than at the surface. It must be remembered, however, that the last Ice Age scoured off a good many feet from the top of these lodes, and that the present rock surface is quite likely a surface which was several hundred feet down before the last Ice Age, so that not infrequently we find rock relatively fresh and unaltered close to the surface. Now, the cost of exploration of a lode through a covering of drift increases very rapidly with the thickness of the same. Not only is all the drilling work through the drift so much dead work, but the cost of shaft sinking and handling water and every other cost increases markedly with the heavy drift covering, and we know enough to realize that in many places this covering is several hundred feet thick. We also know, as has been said, that the copper occurs in a porous bed. This would be most easily eroded by the glacier of the Ice Age, and is likely, on the whole, to be low and to be covered over, while the hard, compact rock, like the ridge of the Cliff, stands out, but, as is known, is not copper-bearing.

Thus surface work by the geologist in regions largely drift covered and on amygdaloid outcrops, which have been leached, can not prove no copper, though he may point out that it will be expensive to locate and he can point out where the moraines cross the range and the drift will be thick‡.

‡F. Leverett, Pub. 7, Pub. 6, pp. 48 and 525 of Michigan Geological Survey.

Two or three other considerations should not be overlooked in discussing the completeness with which the Copper Range has been explored. On the southeast, it is bordered by a great fault. The uplift here, we are all agreed, began before the Keweenawan formation was by any means complete. In fact, it very probably marks the great crack in the crust of the earth along which these enormous floods of lava were outpoured. This is indicated by the occasional appearance of deep-seated intrusives along it. The earlier beds, therefore, were eroded and furnished their pebbles and debris to the later beds and, in so far as they contain copper, this too may have been collected in these later beds.

We find, for instance, pebbles of coarse quartz porphyry just above Lake Linden in the Calumet conglomerate, and we find pebbles of agate in some of the conglomerates, evidently derived from the lower traps. But these disturbances did not cease even after the Keweenawan formation was complete and the copper lodes formed. Toward the close of the Keweenawan, or, as some would make it, later, the earlier, uplifted beds were overlapped by the sandstone, which I have called the Jacobsville sandstone—the Eastern sandstone. But this Eastern sandstone itself was slightly disturbed, as we see around Limestone Mountain, where later beds of the Paleozoic are folded up. Thus we find Eastern sandstone near the contact line dipping in various directions and distorted. It appears that the Keweenawan Range has been again uplifted, if our drill holes have not misled us (and I do not think they have), for we find the Eastern sandstone overridden by the Keweenawan formation, and the drill holes will pass from the traps to the Eastern sandstone and back into the traps which the Eastern sandstone overlaid.

Now the Baltic lode lies not very far from this contact line. Many geologists would lay great stress upon such faulting disturbances and their effect in accumulating copper.

So far as my experience goes, where there is a great deal of disturbance and the rock is throughly decomposed, it is

not likely to contain copper. The copper seems to be driven from it into a zone between the thoroughly disturbed and the less disturbed rock. This is important. One should watch the foot and hanging of pervious belts and the shattered zones between a heavy impervious bed like the Greenstone and the main pervious and shattered belts. I suspect attention may have been too narrowly confined to the amygdaloid itself. It would, however, seem quite fairly likely that such great disturbances, as we know exist along the contact line of the Eastern sandstone, should have some effect in accumulating copper not far off. Such appears to be the case with the mines of the Copper Range Company.

Now, if we ask what amount of exploration has been done along this line, we find that outside the most superficial examinations, it has been tested in perhaps twenty-five or thirty miles out of the course in Michigan of over one hundred.

The geologists of New Jersey, in considering the origin of the copper, which occurs in small quantities there, have laid great stress upon intrusives beneath. We find just north of Bessemer an intrusive sill of gabbro and yet there has been practically no exploring done of any importance west of the Porcupine Mountains, though the dips there are high and there has been no known geological reason why copper might not occur, nor why there might not be copper on the other side of the Porcupines in the synclinal symmetrical to that where the White Pine is.

What does the examination of geologists represented by Publication 6 and Monograph 52 amount to? This is a fair question. I wish to emphasize the fact that such examination by no means proves that the area covered does not contain copper in commercial quantities. The region of the Lake mine was, for instance, visited a number of times by geologists and there was no especial sign on the surface that an important commercial deposit existed below. The most that can be said is that the examination of the geologists have made it rather unlikely that any valuable commercial deposits out-

crop plainly at the surface. But, then, the examination of the geologists has shown that in most cases the outcrops are of the hard, compact beds and not of the softer beds where copper is most likely to occur. The examination of the geologists has also shown that the Copper Range, in many cases, probably extends under regions which are covered with anywhere from thirty to several hundred feet of superficial deposits, left in the Ice Age. It has also pointed out certain great ridges of compact rock, like the main felsite area of the Porcupine Mountains and the Greenstone ridge, where prospects are relatively unfavorable, and it has enabled us to locate in a general way the course of the lodes where exploration is relatively favorable. But only a small portion of these lodes, as we have above seen, has been tested, even in the districts which are highly accessible and favorably situated. In many cases they are blocks of land which, for personal reasons, have not been developed. Who would dare say, for instance, that the two square miles so close to Calumet, owned by the Torch Lake Mining Company, might not contain as good prospects as those recently touched by the drills of the Mayflower and Old Colony or that the Naumkeag nearby may not be a second Quincy? When we consider that west of the Victoria (that is to say, for about one-third of the longitude) there has been but practically one small district at all carefully explored, that from the Victoria to Houghton little or nothing has been done on the upper side of the range and that north of Gratiot River very little has been done on the south side of the range, it would be not too much to say that not five per cent of the range has been thoroughly explored.

But the work of exploration of the future will now be more and more expensive, because it may fairly be said that the easiest regions to develop have already largely been skimmed over, and we need not expect to find mines staring us in the face—they must be sought with long and expensive and deep preliminary explorations, and even when found,

there will be a heavy expense which may sometimes be enough to retard the development of a worthy exploration for years. These preliminary costs ought to be charged against the cost of copper.

To sum up, among the things which seem worth doing in the future is the testing of the Nonesuch horizon, which could be done by churn drill, at various points, the testing of the horizons of the upper part of the series from the Atlantic Mine south, the testing of the lodes of the lower part of the horizon, especially from the Mayflower Mine north, as well as extension of the exploration which is going on at various points along the horizon of these lodes at the present time. It must not be forgotten, also, that copper is not necessarily confined to bedded lodes. There is no real reason why fissure veins, such as kept the Central Mine running for years, may not, with numerous impregnations, prove a valuable source of copper. There is no reason why contact between intrusions of felsite or gabbro such as has produced some accumulation of copper at the Mendota, may not elsewhere be of very much greater value. And it is not too hastily to be assumed that the whole west end of the range is valueless, or that the range that goes south of Lake Gogebic will fail to yield something of value, though the dips there are so flat that, if what I have said above is true, it is relatively less promising.

DISCUSSION.

MR. IMHOFF: I would like to ask Dr. Lane the advantages which he expects to derive from the angle across the formation instead of vertical drilling. I think I get his point, I think he intends the idea shall be to get further information on the individual beds, is that right?

DR. LANE: Partly that and partly to test the question of cross fissures and the distribution of copper relative to cross fissures in such cases.

MR. SPERR: Did I understand Dr. Lane to say that no copper ever occurred in impervious beds?

DR. LANE: It is not likely to occur, yes, I may say no copper has ever occurred in commercial quantities in impervious beds. I do not mean by that to say you do not find streaks of copper running through, that is not true, but not in commercial quantities.

MR. SPERR: Then I do not understand clearly what would be defined as an impervious bed.

DR. LANE: By impervious I mean trap and by pervious I mean conglomerate or amygdaloid sandstone.

MR. SPERR: Perhaps I shall have to state a definite example; whether the Ogima lode in Ontonagon county would be considered a pervious bed or an impervious bed.

DR. LANE: I am not familiar with that.

MR. SPERR: In one of the mines it is known as a trap lode and it is the character of rock that I would class, myself, as impervious, and in places it contains copper in commercial quantities. There is some heavy copper, but most of it is in the form of what is known as shot copper, good, washable copper.

DR. LANE: I have not visited the Ogima lode since it was developed, and Nature is always likely to have a card up her sleeve. I should, perhaps, qualify what I said by including in the term "pervious beds" shattered zones or shear zones. You take the example of brecciated fault veins such as you find out west, there you have certainly got a pervious line, but the Ogima, I don't know about.

MR. SPERR: I believe you also stated that in the shattered zones you would not expect copper.

DR. LANE: No, I said "too much shattered" and I emphasized "too much," what is too much is a question, but I can give you an illustration; take the example of the Atlantic Mine on Section 16 as an illustration of what I call too much shattered; generally speaking, there the vein is poorer, and the old working of the Isle Royale Company, on the Baltic lode there is some red fluccan occasionally and boulder copper, but no commercial copper.

MR. SPERR: Referring to Section 16 as a shattered zone, is it not true that the Superior is just as much shattered as Section 16, but in the Superior, north end, the copper does not happen to be leached?

DR. LANE: I am not prepared to say as to the relatively shattered conditions; it is obvious that in part of the Superior the copper has not been leached out, in part of it, it seems to be there.

MR. SPERR: Do you think in the north end the copper has not been leached out and in the south end it has probably been leached?

DR. LANE: I don't think there is very much doubt.

MR. SPERR: If it has been leached out, you may expect to find that copper at greater depth?

DR. LANE: Yes, as long as the water is fresh and as long as the lode is very much shattered, I don't think the case is hopeless.

MR. SPERR: What if it becomes salt?

DR. LANE: As it becomes salt, it is obvious that the leaching has not become thorough, but as one continues down, if you get three kinds of water, fresh, sodium chloride and calcium chloride and it still remains lean, the chances to get copper are, in my judgment, very much diminished; of course you may still have a shoot of copper coming in sidewise along some other line.

MR. SMYTH: Do you ever get this strong calcium chloride water in connection with leached ground?

DR. LANE: Surely.

MR. SMYTH: Has there been leaching there?

DR. LANE: No, my idea, to go back once more, is here; you have these copper bearing beds varying anywhere from ten to thirty thousand feet and they were originally hot, and they stayed hot a good while. As they cooled they would contract but the water in them would contract more, and they would tend to soak up water from the sides and the original water there, which seems to have been of a saline nature,

seems to have worked downward toward the center on the pervious lines, and in a lode it tended to work out towards the sides. That migration would go on in the calcium chloride water and you are liable to have copper, and generally speaking you do have copper if the water is salt, and you may have copper where the water is fresh, but the mineral crest, as it is sometimes called, the place on the particular lode where it is richest is, I think, below the zone of fresh water and somewhere near the top of the zone of salt water. For example, take the Calumet & Hecla; they had fresh water with probably about as rich as it ever was, say 2,000 feet down; you know more about it than I do, perhaps; and it has grown less rich as it has gone down a greater depth and there is still a great deal of copper there. You can easily see, in such a migration—it is not a circulation but a sucking in, absorption, working towards the center—there would be a chance for copper deposit in the salt water zone, only the nearer you get towards the center of the Lake Superior Basin, the less the motion would be and on the whole, the less the accumulation of copper.

MR. CARNAHAN: I should like to ask Dr. Lane what is his opinion as to the possibility of the extension of the Keweenawan series under the Eastern sandstone and the possibility of copper formation in that region.

DR. LANE: I think for a short distance under the Eastern sandstone, in some cases you may find it at a relatively shallow depth, but I think before you have gone a mile or so it will be so deep that it will be practically out of consideration for the present generation of mining.

MR. CARNAHAN: We have in that connection a few islands of trap, we might say, protruding up through the Eastern sandstone and it was with reference to those that my inquiry applied particularly.

DR. LANE: Like Silver Mountain?

MR. CARNAHAN: Yes.

DR. LANE: The United States Geological Survey, in Re-

port 52, is inclined to consider the depositions of copper as cotemporaneous with the Keweenaw series, and I am inclined to agree with that. I see no reason why there may not be a deposition of copper under the blanket of Eastern sandstone and that the copper forming sucking-in took place before the blanket laid down had checked it. The main objection to supposing that there is much copper off to the south around Silver Mountain in my opinion, as the dips are rather flat, and I am inclined to think that the flatter the dip the less the disturbance, the less sucking-in there has been, the less accumulation of copper, so I should not consider the chances of Silver Mountain especially good.

MR. SMYTH: I would like to ask whether the fact has been positively determined that the sandstone becomes thicker east of the fault?

DR. LANE: Just this: On the Torch Lake Mining Company section, by drill holes, after passing through trap they passed through 50 feet of Eastern sandstone and then passed down through the trap; you pass a little further west to Lake Linden and the Calumet & Hecla put down 1,500 feet through sandstone. It was that I had in mind, evidences like that.

MR. KELLY: I would like to ask for some information in regard to the extent of these individual beds. For instance the Calumet conglomerate, how extensive is it? Is it possible to trace it for an indefinite distance, or is it comparatively local; in other words, are the beds continuous this way (indicating by holding two sheets of paper end to end) or are they separate beds, comparatively short laterally?

DR. LANE: In other words are they continuous lines or more like the sardines in a box, is that the idea?

MR. KELLY: Yes.

DR. LANE: I think individual beds, for example, the Greenstone, can be distinctly identified as a particular bed from Portage Lake to near Keweenaw Point and from Portage Lake across to Isle Royale and underneath that in most places comes the Allouez conglomerate; then again you are liable to

have marked changes in a very short distance; take the Calumet conglomerate, at Calumet it is largely made up of quartz porphyry pebbles. As you go south to Portage Lake, by the time you come on the Franklin Junior—in fact, I believe by the time you get to Osceola, it is an entirely different thing, it is mostly made up of trap, so you would hardly recognize it, but as a geological horizon you can still trace it and I can trace it north from Calumet as far as Mandan.

There are some times, as we shall hear later, cases where beds are extremely local. In other words some of them are quite persistent and I find, on the whole the thing I am surest of, is that, if there is a heavy bed of trap it must persist for some distances. Of course, you can easily imagine, if this were an old surface, a desert surface, like the Snake River Valley out in Oregon, you would have deposits here and there in places where there would be just a little wind blown sand.

FOOT WALL SHAFTS IN LAKE SUPERIOR COPPER MINES.

BY L. L. HUBBARD, HOUGHTON, MICH.

This subject may be approached from two closely related standpoints—of economy and of safety. I say related standpoints, for in safety from interruption to regular operation lies also economy. It was mainly from the first standpoint that the writer became interested in the subject, and the aim of this paper is principally to call the attention of members of the Institute to an uneconomic practice, and incidentally to evoke discussion on some other phases of mine development.

In the Lake Superior copper district, the increase in the size of mining units and improvements in mining methods, are constantly tending to reduce costs. Methods in vogue today may be materially modified, if not discarded, some years hence. Any conclusions based on figures of today should, therefore, be scrutinized with reference to possible future improvements. It is also true that no policy of administration nor system of mining will fit all mining properties alike, or necessarily the same mining property during all periods of its activity.

Without pretending to give figures taken actually from cost sheets, I purpose to assume conditions, practices and costs with intent to present a fair statement of the case. You may substitute any other factors for any mine to suit conditions different from those assumed by me.

In accordance with time-honored practice, shafts have usually been sunk directly in the lode to be mined, because in and from such shafts a thorough examination of the lode is possible in the quickest and most economical way, and the

average richness and physical condition of the ground are directly ascertainable. Where, however, these considerations are not of paramount importance, as for example possibly on a property where one or more shafts are already down, and others are to be sunk to increase production from the same lode, or where by diamond drill exploration or otherwise, the lode is known to be rich in the neighborhood of the proposed shaft or shafts, we may well consider an alternative to a practice that ties up in shaft pillars a considerable amount of values during the life of the mine. Moreover where the dip of the lode is variable, as is often the case, or where the average dip is not known in advance, the exploratory function of the lode-shaft is at least curtailed, if not altogether lost, and while the dip of the shaft can eventually be readjusted to the actual dip of the lode, a variable dip in the shaft is from an operating point of view, at least a detriment, if not a menace.

A shaft sunk in and within the limits of the vein or lode, must be supported on at least three sides, to its ultimate depth, by an envelope of lode-rock capable of sustaining not only its own weight, but also the pressure induced in the overlying rock by the removal of its support under a large area. Under one system of mining, supporting pillars are left from point to point in the levels or stopes, if possible only where the rock is not worth extracting; under another system, good rock removed is replaced by a filling of waste rock, but even if the filling be complete, a settling of at least ten per cent of the filled area has been declared to be possible; under the first system even more disastrous results have followed, from the crushing of pillars and caving of the hanging. In either case movements in the shaft are apt to cause delay and expense, if indeed the shaft be not ultimately wrecked. I need not cite examples of each kind from our immediate neighborhood.

What then are adequate shaft pillars will doubtless depend in each case, much upon the dip of the lode, the inherent

strength of the rock, and the absence of "slips" or joint-planes. The size of the pillars can hardly be stated in terms of an equation. Based on local practice, a minimum average width of 50 feet in the pillars for a shaft intended for a depth of five thousand feet, seems small enough, and a mine manager would run little risk of criticism if the width were even greater. If the dimensions of the shaft be assumed as 10 feet by 20 feet, and the lode be 15 feet thick, the block of lode material surrounding the shaft will then have 120x15, or 1800 square feet of area, less the shaft opening, (10 ft. by 20 ft.) or 1600 square feet net. We may neglect the small deduction for the shaft plat and for drifts or crosscuts through the pillar at each level, for there are few lodes in this district that will not average more than 15 feet in thickness.

Let us now determine the probable life of the better class of Lake Superior copper mines, by considering the possible annual extraction: and here again we may note that much larger daily tonnages are likely to be hoisted in the future than have been hoisted in the past—by companies that own large acreages, and have at their command ample stamping facilities and other adjuncts, which smaller companies cannot afford, or do not feel warranted in providing.

Shafts in the exploratory period are, for obvious reasons, usually equipped with small hoists of capacity limited both as to depth and tonnage. As openings in the lode are extended and permit of larger production, heavier hoists with larger drums are installed, so that the rate of extraction can easily be increased to the limits imposed by the other parts of the system; at least to within a comparatively short time of the exhaustion of the mine. Mr. Uren has shown that a hoist operating a six-ton skip from a depth, I believe, of 1600 feet, can, if required, actually hoist from a pocket at the rate of 1800 tons per shift.

From official reports, I find that production in this district in recent years in mines using skips of a capacity up to six tons, has varied from 600 tons to 700 tons, and in one

case has reached about 1,000 tons per day per shaft. For purposes of our computation, let us assume 800 tons per day as an average product, and the length of the drifts as 800 feet on each side of the shaft. You will readily see that this latter factor is elastic, for you can increase your tonnage at will by lengthening your drifts, and will not thereby shorten the time of extraction to the ultimate depth of any one shaft. In other words the length of the drifts is, under an economic tramming system, quite immaterial, except in so far as it affects the distance apart of the shafts, that is, the number of shafts that may be required for any particular mine. Our immediate consideration is the probable costliness of one lode-shaft.

I have in mind one mine where if we allow one thousand feet beyond each of the end shafts, the average tramming distance on each side of the shafts is 675 feet. The average annual rate of advance for the shafts for several years has been about 175 feet, which, I suppose, represents about the annual rate of vertical extraction in the mine.

If we allow 1600 feet of drifts per level to a shaft in a lode 15 feet thick, the cubic contents of the lode within those limits and between two levels 100 feet apart are 1600x100x15, 2,400,000 cubic feet; less shaft pillars, 160,000 cubic feet; total lode contents, 2,240,000 cubic feet. At 12 cubic feet per ton, this is equivalent to 186,667 tons, of which say 60 per cent is merchantable rock, or 112,000 tons. At 800 tons per day, this would last 140 days and would be equal to an extraction of one foot in depth along the edge of our mineral block in 1.4 days, or 222 feet in a year of 310 days.

At this rate the life of a shaft 5,000 feet deep would be 22 years. Taking the example quoted in a preceding paragraph, and adjusting the lode area to the different length of drifts, the necessary rate of vertical extraction would be (675:800=x:175) 148 feet per year, which is about two thirds of the depth required under the figures assumed above.

This would make the life of this mine, by comparison, about 33 years.

Having assumed certain conditions under which the life of a shaft five thousand feet deep would be 22 years, it now remains for us to calculate the value of the mineral in the shaft pillars, and the probable profit dormant in them. The average price of copper for a period of thirty years past has been computed, I believe, at 14 cents*. Several of our local mines working in so-called amygdaloidal deposits, have been and are producing copper at prices between 7½ cents and 10 cents a pound. Companies that have facilities to extract and treat the greater tonnages figured in the foregoing computations, it is fair to assume, could easily produce copper at 9 cents, or at a profit of 5 cents per pound on the average price above quoted. For convenience, let us assume 20 pounds of refined copper recovered per ton of rock stamped. These figures are exceeded by a goodly number of our best mines, one of them over 20 years old and working at a depth of probably 3,500 feet, but it is likely that in most of our mines this average may not prevail to a depth of 5,000 feet. The pillars in 5,000 feet of shaft that have a net area of 1,600 square feet will thus contain 8,000,000 cubic feet or 666,667 tons of rock, which at the average per cent. of merchantable rock in the lode should yield 400,000 tons of rock containing 8,000,000 pounds of copper. This represents a clear profit of $400,000, and this rock, if the shaft were in the foot wall, could be extracted more cheaply than any other rock in the mine, and much more cheaply than shaft pillars can ever be extracted under the most favorable conditions. However confident we may be that these values will all be eventually recovered, we are still facing the large loss entailed by tying up $400,000 for twenty years. One dollar a year for each of twenty successive years, with interest at 3 per cent. compounded, will amount to $27.67. Each year during the

*The Wolverine Copper Mining Company is said to have received 15.18 cents per pound for its copper during the twenty-two years of its existence.

life of the mine there is wasted in each lode-shaft, in the form of dead capital, an amount of money which in 20 years would aggregate $153,000. If by any mishap we should lose our shaft pillars, our total loss would amount to over half a million dollars.

At first glance it may be thought that a shaft in the foot wall will be more costly and entail more delay than one in the lode, by reason of the crosscut necessary at each level. If the shaft is located about 50 or 60 feet from the lode, it will probably not be endangered by the blasting in the lode. At the end of the 60 feet of cross-cut economical mining may begin at once, in two directions, whereas with shaft pillars extending 50 feet from the ends of the shaft, at least 100 feet of drifting is necessary before reaching those parts of the lode that can be similarly mined. It is true on the other hand that rock extracted in the drifts may pay the whole or at least a part of the cost of drifting. It may also be true that the foot-wall rock may be harder and more costly to mine than the lode-rock, and this is one of the few economic disadvantages of foot-wall shafts.

It has been suggested to me by a prominent Houghton engineer, that in a mine where the rock appears to be particularly under stress, another advantage in a foot-wall shaft would lie in the fact that as soon as the lode matter is extracted opposite the shaft, the stresses in the rock next to the shaft are immediately relieved, and there will result a greater stability in the shaft on this account. I believe this point is well taken, but I shall leave its discussion to you.

In any event, if the shaft is in the foot wall, it will be more exempt from those disasters that sometimes overtake mines even where the greatest foresight has been exercised. With the shaft intact, safe egress for the miners that can reach the shaft is assured, and ingress to any level in the mine can speedily be obtained. With a shaft out of commission by reason of earth-stress, production is interrupted, costs of production are raised, and shaft repairs made necessary at

considerable cost. For these several reasons will not the physical advantages of a foot-wall shaft fairly outweigh the slight probable disadvantages and miscellaneous extra costs, even if we disregard entirely the enormous capital locked up in shaft pillars? I have believed in the theory of foot-wall shafts for several years, and first put it in practice at the King Philip mine in 1906, and later at Winona and Ojibway. I am free to say that at the latter location, the increase in the thickness of the foot wall bed, and the unexpected flattening in the dip, have contributed to make the cross-cuts from one of the shafts a good deal longer than was anticipated; but, on the other hand, each shaft has been sunk nearly 2,000 feet, of which only about 200 feet has required timber in one shaft, and 300 feet in the other.

DISCUSSION.

MR. DENTON: Dr. Hubbard's paper opens up a very large subject and a very important one, and a very difficult one to have positive opinions upon; and if there are any others here who can offer suggestions, from experience or otherwise, we shall be very glad to hear them.

MR. SPERR: I would like to hear that question discussed, I am sure it is not very late. There must be some one here that has an opinion on it. I do not wish to express my opinion now. (Laughter.)

MR. LAWTON: I am inclined to think that, if any of us could conceive of a copper bearing lode in this district without limits and carrying twenty pounds of extractable copper per ton of rock, the location of the shaft would not make much difference in the final results; but, in starting a shaft for prospecting purposes in fairly unknown territory with the dip between vertical and fifty degrees, and with limited means, I am inclined to think one might save the cost of those pillars just mentioned, by the value of the copper in the rock taken out in sinking the shafts in the lode, and by the copper in the rock won from the drifts. If there is less than fifty degrees incline, I should be inclined to favor the location of

hoisting. Where the dip is less than 50 degrees, the cross-cuts would be excessively long, but even in such cases I have heard the opinion expressed that it would be better to sink vertical shafts on account of the cheaper maintenance.

DR. LANE: What has been the experience of foot-wall shafts on the Gogebic Range?

MR. DENTON: Will some gentleman from Gogebic answer that question?

MR. HEARDING: In regard to foot-wall shafts, they have found that they have had some difficulty in keeping them up when the shaft was not located far enough away from the dividing line between the slates of the foot-wall and the foot-wall itself. The foot-wall proper had a tendency to ride down. If it is far enough back where that tendency to cleavage is done away with, that difficulty would be obviated.

MR. DUNCAN: I would like to say if that shaft had been put back into the slates, I don't think they would have had any difficulty with it.

MR. HEARDING: If it had been clear back in the slates, yes.

MR. COLE: In sinking shafts, the problem of finding suitable rock in which to make such opening that will always insure a safe and permanent outlet for the handling of our men and material necessitates most careful study. I would say that after many years of work of rehabilitating mines where we found many shafts sunk in the ore deposits, considerable difficulty was experienced in finding a place for the outlet that would insure permanency.

On the Gogebic Range we find it necessary at times to drive cross-cuts into the foot-wall rocks for a distance of over two hundred feet; we found that the quartzite underlying the ore deposits is apt to move; I cannot explain why it moves, but I assume it is overlying pressure has something to do with it, possibly hydraulic pressure from water underneath might have had a lifting effect after the ore bodies were removed; the slate underlying the quartzite were friable and

in many places when exposed to the air they would begin to swell. I think possibly with a lining of concrete that feature would be overcome, but in one case, I remember, one of the shafts sunk on the Pabst property, at a point 250 feet in the foot-wall slates, we had difficulty with swelling ground, caused by the friable nature of the slates, and we had to abandon the plan of supporting the walls of the shaft with steel frames and had recourse to putting timber in with sets placed very close together so we could control the crushing effect. In the iron mining districts we were all inclined to favor foot-walls for our permanent openings or shafts, because we generally found the rocks were more stable than any of the hanging-wall rocks, and I believe, over a term of years, we had very good success in locating shafts that bid fair to last as long as the ore bodies will be mined.

Coming to this district, with which I have had some work to do and some familiarity with the conditions to be met, I would say that in support of Dr. Hubbard's view, I believe that the conditions in each property have a good deal to do with the question of where your shafts should be located. In the first place, on the Calumet Conglomerate at the point where intense mineralization occurs, the foot-wall rocks are not stable; a great many years ago they sought to find a point where the rocks would be stable in which to sink permanent shafts, on account of the difficulty of supporting by pillars the inclined shafts that were sunk on the lode. On an investigation conducted by driving a series of crosscuts with which Mr. Grierson is familiar, the fact was disclosed that foot-wall rocks were not stable and that it was not a place to sink shafts. When the question came up as to the proper location of shafts for the Ojibway Mine, in spite of the fact that I was disposed to favor the location of permanent outlets in the foot-wall rocks, I rather took issue with Dr. Hubbard as to the proper point in which to sink the shafts. Of course the man who wants to explore the rocks naturally would like to have every foot of devel-

opment work in the lode itself, so as to be able to determine as quickly as may be whether or not it will pay. And further, I was familiar with the fact that underlying the Calumet Conglomerate were rocks that were not stable, so that matter was discussed as to where the best location would be to locate the shafts, and he ascertained that the trap rock underlying the Kearsarge lode was fairly stable. At many points openings had been made which disclosed the fact that the rocks were good and strong; therefore, we finally agreed that the shafts should go down in the foot-wall rocks, and on any property where the lode is apt to be a good, paying one and the foot-wall rocks are stable, I believe in the copper mining district it is the place for such a shaft. It will be permanent and it can be sunk, as far as the actual breaking of rock is concerned, with very little more cost. When you have your shaft in good, stable rocks underneath the lode, you are in position to form good chutes in which your rock will be dumped and discharged into your skips affording economical handling. Above all, the main feature, once you get down in your mine 2,000, 2,500, 3,000 or 4,000 feet with your shaft located back in good foot-wall rocks, you are relieved from the element of uncertainty as to maintaining the opening. You have had instances of what it means to have shafts located in the lode and supported only by meager pillars, you may say, on either side of a shaft, and the hanging wall giving such an enormous pressure that the pillars are affected and the absolute safety of the outlet is questionable. Therefore, I think that the very important feature is that if you have got a paying lode and you have got stable underlying rocks, you had better have your shaft back in that foot-wall rock; you haven't then the element of uncertainty hanging over you all the time as to whether that shaft can be maintained for a permanent and safe outlet for handling your men, as well as for the conduct of your mining operations.

MR. BUSH: As to Mr. Sperr's remarks in regard to ver-

tical shafts and Mr. Cole's remarks on the Gogebic Range, a mining company has commenced to sink a new shaft on the Gogebic where the dip is 67 or 68 degrees, and they plan on putting down a vertical shaft which will mean a cross-cut of 700 feet or more, and they figure from the experience they have had that they can afford to sink a vertical shaft and drive these cross-cuts and save money as compared with the maintenance of an incline shaft.

MR. COLE: I would like to ask Mr. Grierson as to the stability of the foot-wall rocks under the Calumet Conglomerate.

MR. GRIERSON: From my experience, I would suggest that, if you are going to have a foot-wall shaft, don't wait until you have a shaft in the lode, and then try to put another shaft under it. That won't work. In the case referred to by Mr. Cole, about 25 years ago, our No. 2 Calumet shaft hanging began to get heavy and we decided to raise a new shaft in the foot-wall and under the pillar protecting No 2 shaft. We ran cross-cuts into the foot trap at the 20th and 21st levels, about 60 feet from the conglomerate lode, and began to raise for the new shaft. I was doing the mine surveying then, and remember the difficulty we had in keeping our line and dip plugs in place. While the trap seemed to be hard and firm, it would crack and "spit" so that it was unsafe for men to work under it, and we had to give it up. We afterwards sank our "slope shaft" below the 57th level, in about the same relation to the conglomerate lode and had no such difficulty. Of course in the latter case the conglomerate above the foot-wall shaft was intact and undisturbed.

Referring to Professor Sperr's remarks about vertical instead of inclined shafts, I have not had experience with lodes dipping over 50 degrees, the formation at Calumet having a very uniform dip of about 38 degrees. We must remember that there is a difference in depth of shafts needed in the Iron and Copper districts. Here our shafts must

be from three to five thousand feet deep. To equalize the length of cross-cuts your shaft will have to go through the lode you are working. For instance; our Red Jacket vertical shaft went through the conglomerate lode at about 3,300 feet from surface, and went down to 4,900 feet. The only trouble we have had is where the shaft went through the lode, even though we left a pillar 500 feet square to protect the shaft. I think if you know that you have a lode carrying a working value of copper, and the question is simply getting that out and taking 25 or 30 years to do it, then the foot-wall shaft would be all right, provided you have the right material to sink the foot-wall shaft in, but if you are starting a new mine and haven't all the money you might have, and want to know something about what is ahead of you, both in sinking and drifting, it is quite an advantage to have your shaft in the lode. It is a more comfortable feeling to the man in charge, than if he is working in the dark, not seeing what is just above him in the lode.

MR. DENTON: I think that the authorities on the direction of the overlying pressure are pretty clear, for they say that that direction is somewhere between a vertical line and a line at right angles to the dip. Now that pressure is infinite compared with our power to resist it, and I think that experience has shown, up here as well as elsewhere, that the character of the beds immediately above and below the one which contains the mine affects the results produced by these great stresses. We may have the distinct parting planes formed by interbedded conglomerates as at the Atlantic; or large clay seams formed by the ancient movement of the adjacent rock masses. Such clay slips occur in all the amygdaloids to varying extent; in some cases extending over a large area of the mine parallel to strike and dip; in some not quite parallel to the formation; in some at right angles to the plan of the lode and intersecting.

The question hard to answer, therefore, is how far in the foot-wall must the shaft be to remain unaffected by this

overlying stress which must find relief in some manner. There are some examples where the shaft pillars are being slowly compressed, allowing the hanging-wall to settle, and others where the pillars seem strong enough to remain intact but are forced into the foot-wall, causing the latter to bulge and break, giving an effect that may be illustrated by sticking the finger into a mass of clay. In no manner does it seem possible yet to mine so that this overlying burden can be transferred from the deposit being mined to the underlying bed in such a regular and uniform manner as not to affect the shafts somewhat.

MR. LAWTON: I think our chairman put it well at the start, it opens up a very large question. There are different views; opinions differ, but the resultant location of the shaft should be governed, I should judge, by the particular, or individual conditions, rather than by any general law. There are questions in this district of extreme hanging-wall pressures being transmitted to the foot-wall, which have to be taken into consideration to a very, very great degree.

Answering Mr. Sperr, I should say it is not always wise, it may not always be wise, to extract the lode from the hanging of a shaft that is sunk parallel in the foot-wall of the lode and dips less than fifty degrees. It depends largely on the distance of the shaft from the lode into the foot-wall. With regard to the question in general, I should say there is a difference between the Copper Country and the Iron Country, a marked difference, and a marked local difference in the Copper Country itself in the conditions surrounding the lodes that are being extracted.

I may add as a note that there are five connected shafts in the Copper Country of Lake Superior that long years ago were sunk in the lode; and since then, for fifty years or more, they have been sinking in the lode. The miners have always had a safe ingress and egress to the mine. Those shafts paid dividends early in their career; they have been paying dividends now for over fifty years, and are still paying dividends.

It may be possible that foot-wall shafts are better; but I do not know of any shafts sunk in the foot-wall parallel to the lode that can be used to compare with these.

MR. SPERR: I can't see how any weight can be added to the shaft by reason of taking the lode out from over it, that is what bothers me.

MR. LAWTON: It does me also.

MR SPERR: There the individual experience is valuable. I think in the original Quincy mine there is a thrust from the foot-wall side. I have thought that this was due to the natural stresses found in stratified rocks, but am lately more inclined to the opinion that it is due to pillars left in the body of the mine away from the shafts. May not rock flowage be accountable for many of the phenomena of rock movement in that mine?

In the early days of mining in this district there were many excuses for resorting to the pillar for permanent support. That the excavations would ever attain their present extent was not anticipated. With hammer and drill and black powder it was expensive to break any extra poor ground, and other considerations might be enumerated which made the practice of leaving permanent pillars seem desirable. However, if excavations are to be extended indefinitely and carried on for an indefinite time, pillars may be made use of temporarily, but should not be left permanently. The system of mining should not only provide for the ultimate extraction of the pillars, but also for their early removal or destruction. Even if the pillars are poor ground, they should be broken down under control; or else they will ultimately break down beyond control. If the areas of poor ground are so large that they do not partake of the nature of pillars, it is a different question altogether; and if the pillars are so small that they will crush without doing any harm, it may not be necessary to pay any attention to them after they have served their purpose. In a mine where pillars have been left which can not now be taken out, it may not be desirable to mine out

the lode from over the shaft, because it would probably result in the shaft being thrown upward from the foot. But this action is obviously not due to the addition of weight to the shaft, but rather to taking weight off.

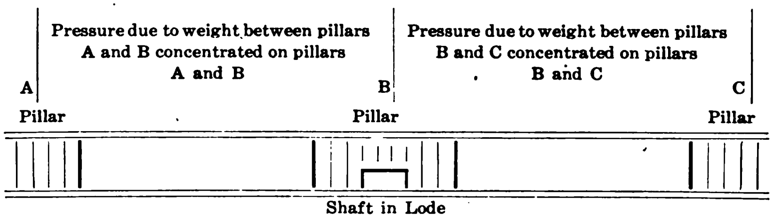
MR. HEARDING: I would like to inquire if there has ever been any work done along the line of the foot-wall or in the rear of it, as you might say, in any of the old mines where the hanging has come down and been lying on the foot-wall, to see if there was any shattering of the rock due to pressure from the hanging.

MR. GRIERSON: The Calumet & Hecla has drifted some 4,000 feet in an amygdaloid lode lying about 125 feet under, and parallel with, the conglomerate lode for water storage levels, and has run a number of cross-cuts between the two lodes at depths varying from 3,000 to 6,000 feet on the dip. In only a few cases have any extraordinary movement of the rock been noticed, and these seem to be zones of great pressure, or unusual weakness, extending across the formation.

CONTRIBUTED BY R. R. SEEBER.

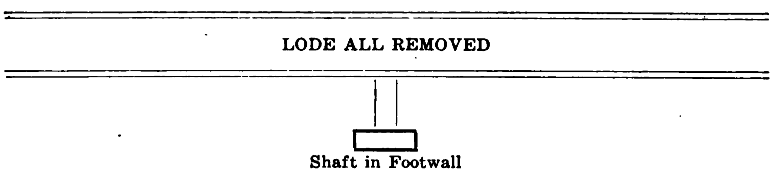
Of course, pressure transmitted from the hanging through the pillars to the foot-wall, sets up strains in the foot-wall and may cause heaving of the foot, most likely at points about half way between the pillars, although possibly at other points, because of local weaknesses. If the foot-wall were particularly weak, swelling of the foot might be caused immediately in the foot-wall side of the shaft in the lode due to stresses set up by the loads on the two shaft pillars. However, the distance being so comparatively short, I do not think that this is likely to occur. With the shaft in the foot-wall and the lode matter all removed, the stresses can be transmitted evenly from the hanging to the foot-wall either by filling or by caving the hanging. The stress then returns to the normal stress which existed before the lode matter was removed. In the process of removing all the lode, if pillars were left at an equal distance from the shaft on either side, on which was concentrated a large amount of load, strains would be

induced in the foot-wall reaching a maximum in the neighbor-hood of the foot-wall shaft. This condition should not be allowed to arise with a carefully considered system of mining.



Pillar B carries total hanging load of distance A to B where A B and B C are equal.

Or normal load is increased in ratio $\dfrac{\text{distance A B}}{\text{width pillar B}}$



Lode matter all removed and space filled or hanging caved thereby removing strain.

BALANCING ROCK CRUSHERS.

BY O. P. HOOD, PITTSBURG, PA.

The matter of this paper was published about four years ago, and the device was also patented but it has not been called to the attention of mining men who are interested in rock crushers. The rock crusher with a vibrating jaw has been developed to such a size that the unbalanced horizontal forces are of considerable magnitude when the crusher is running at a reasonable speed. It is not an uncommon sight to see such crushers working loose in good foundations, or seriously jarring buildings in which they may be placed. It is frequently impossible to run such machines at the speeds recommended by the builders because of the excessive vibration produced. This condition has been met sometimes by expensive bracing of buildings, although it would seem better to remove the cause by properly balancing the crushers.

Where several crushers are on the same floor it has been suggested that they be so arranged as to position or relative speeds that each tends to counteract the vibration of the others, but unless geared together so as to maintain an invariable phase relationship these attempts are futile. In one rock house having three crushers side by side running at slightly different speeds, it was apparent at times that the vibrations were neutralized, but equally apparent at other times all three crushers were in step and producing vibration of the building equal to the sum of the three efforts. The horizontal force producing these vibrations may be two to three tons for each crusher, so that when they are in phase it is no wonder the building vibrates in answer to a force of six to nine tons applied 175 times per minute.

This paper describes the results obtained in counterbalancing a 36x24 crusher, and also gives the method devised for determining the proper counter weight.

The crusher was mounted on a truck resting on rails in such a manner that when the crusher was run by a vertical engine, carried on top of the crusher, the whole device of engine, crusher and car could vibrate freely on the rails. When the crusher was run at fifty revolutions per minute the vibration of the whole mass, weighing about forty-five tons, was nearly one-half inch, 45/100 to be exact. Since, in the manufacture of such machines, it is not the practice to balance the fly wheels, these were removed and carefully balanced so that the vibration could not be due to inaccurate fly wheels. When running at fifty revolutions per minute the vibration was then found to be increased to 48/100 of an inch, showing that the position of the unbalanced mass in the fly wheels had partly counteracted the natural vibration due to the jaw. There was then placed within the rim of the fly wheel certain counterbalancing weights which had been computed. The crusher was then found to vibrate but 3/100 of an inch. Reducing the free vibration from 48/100 of an inch to 3/100 of an inch can be called a practical balancing of the rock crusher, and this is done by the proper placing of a single weight in the fly wheel. It is, therefore, possible to so balance jaw rock crushers that they can be run at any reasonable speed without the vibration of buildings or the destruction of foundations. This method has been applied to old crushers already installed and the vibration eliminated.

In analyzing this balance problem a graphical method was devised which can be applied to many similar problems involving vibrating masses.

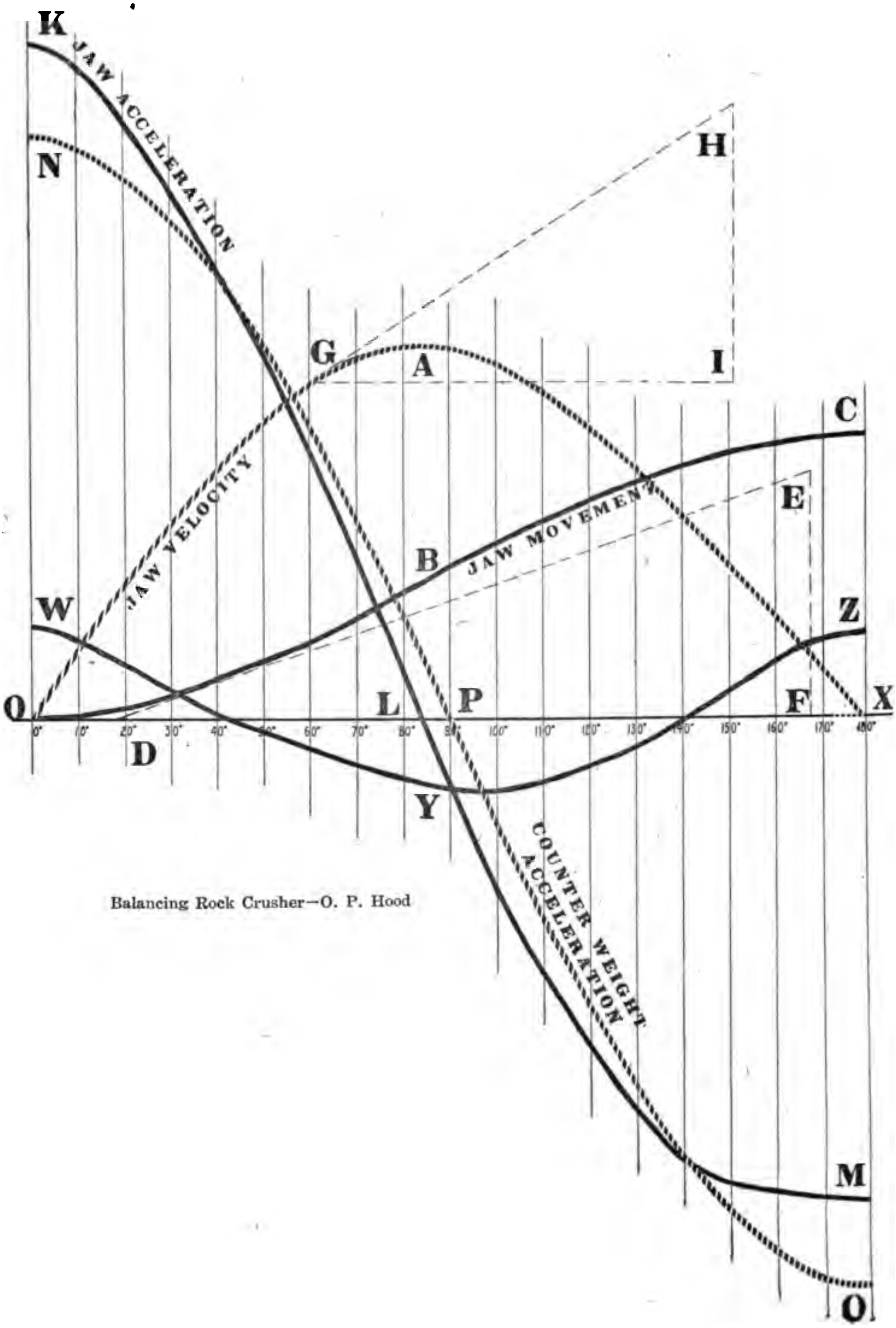
The jaw of the crusher is moved forward by the raising of a pitman operated by an eccentric. The pitman spreads the ends of two inclined links, thus moving the jaw. The eccentric has a uniform rotative motion, so that equal angles are passed in equal times.

This train of mechanism is easily measured and reproduced on paper—and the successive positions of the jaw plotted for the different positions of the eccentric. This is shown by the distance of the curve OBC from the horizontal line OX. Distances along OX represent angular motion of the eccentric, and, since this motion is uniform, represent time. When the eccentric has moved 90 degrees from its lowest point the jaw movement is represented by PB, and when it has moved 180 degrees the jaw movement is represented by XC.

From this curve of jaw movement the velocity of the jaw at any instant can be found. Thus, if a tangent DE be drawn to the curve the jaw would have moved to distance FE in the time represented by DF if the jaw velocity has been uniform. Plotting the jaw velocities obtained in this way the curve OAX is obtained.

The rate at which the velocity of a mass is changing is a measure of the force acting on that mass, so that if we can obtain the rate at which the curve OAX is changing its direction we have a representation of the forces acting on the mass. This can be obtained also by drawing tangents to the velocity curve as at G, where, had the velocity continued to change uniformly at the same rate for the time GI the accelerations would have been HI. Plotting these several accelerations the curve KLM is obtained, which represents to some suitable scale the forces acting on the vibrating jaw necessary to produce its peculiar kind of vibration.

This curve looks very much like the curve that would result from an unbalanced rotating mass, except that it is unsymmetrical, having a greater value at zero degrees than at 180 degrees. If an unbalanced rotating mass were added to the system which would have an acceleration as much too small at zero degrees as it was too large at 180 degrees these forces would be represented by the curve NPQ. These forces could be put in opposite phase to those moving the jaw, so that they would form a counterbalance. The lack of agreement of the curves K and N would leave a small resultant, shown by the



Balancing Rock Crusher—O. P. Hood

curve WYZ, as the remaining unbalanced forces. These remaining unbalanced forces are but a small per cent of the original forces, and their rate of alternation has been doubled. Doubling the rate of alternation would reduce the vibration to one-quarter of the original amount were there no reduction in the magnitude of the forces. It is interesting to note that the forces shown by the curve WYZ can be balanced by a suitable weight revolving at twice the speed of the main shaft. While this is possible it is neither desirable or necessary on a rock crusher because the single weight produces a practical balance. This method of determining unbalanced forces is applicable to many mechanisms and its application might lead to the smoother running of many machines.

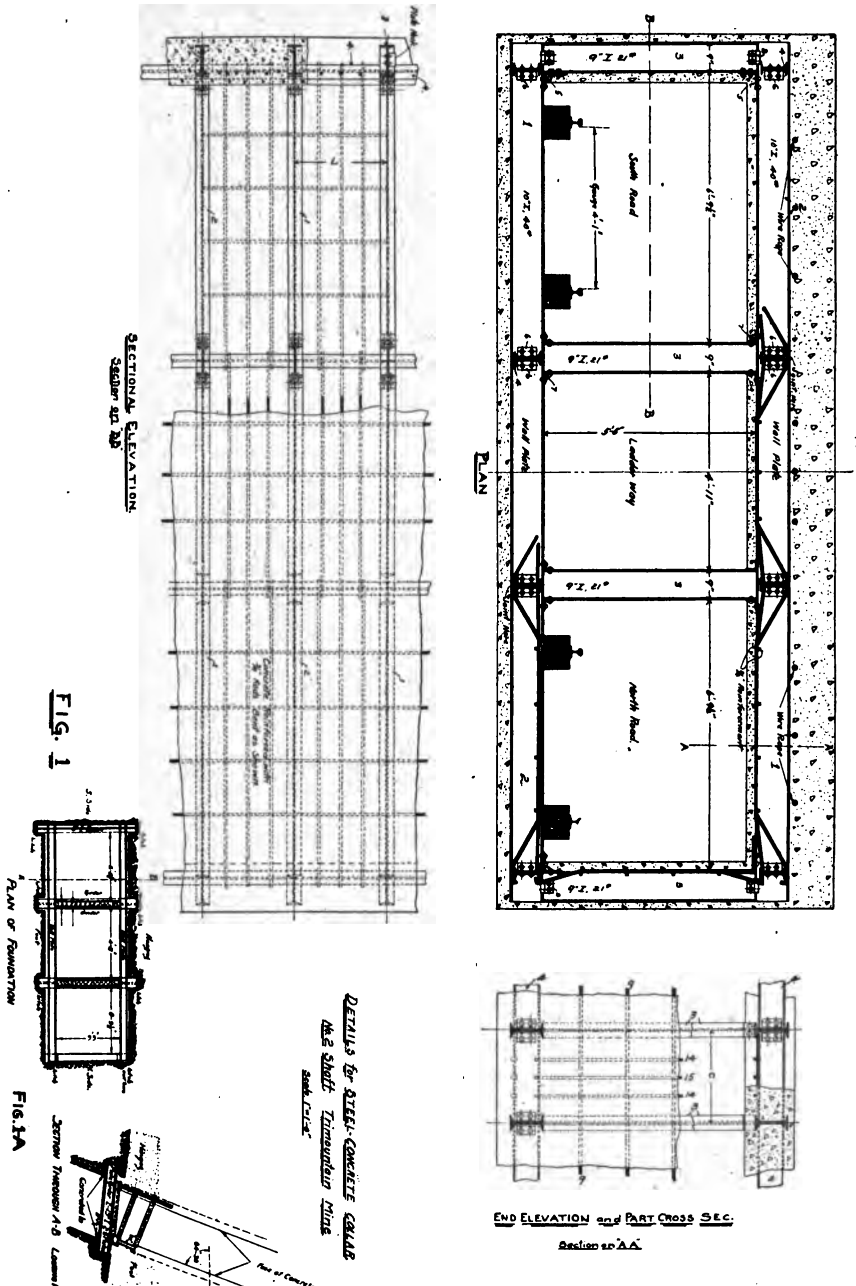
SOME APPLICATIONS OF CONCRETE UNDERGROUND.

BY H. T. MERCER, PAINESDALE, MICH.

The rapid growth in favor of concrete for certain classes of construction has been one of the most noteworthy engineering developments of late years; and in this the applications made to the mining field have played an important part. This is owing to the decreasing supply of suitable timber and to the limited life of even the best timber when exposed to underground conditions.

Concrete has been used for many years in building underground dams, bulkheads, etc., some notable examples of which can be seen at the Chapin Iron Mine at Iron Mountain, Mich. The principal uses of concrete in mines, however, is in connection with shaft support, and it is the purpose of this paper to describe some of the work that has been done along these lines in the Copper Country. Good examples of concrete shaft collars can be seen at many of the mines, and although the details vary somewhat, a description of one or two will perhaps suffice to illustrate this form of construction.

At the Trimountain Mine it was decided to replace the old timber collars with concrete, and work was begun at No. 2 shaft, where the overburden was 80 feet deep, consisting for the most part of sand, with more or less clay and some boulders. To guard against any possible "running" of the sand, and to make the operation of the shaft during construction easier, as well as to reinforce the concrete, it was decided to replace the timber with steel I beam sets, and then concrete between and around the steel sets. The sets would provide a support in case it became necessary to put in lagging to hold



DETAILS for STEEL-CONCRETE COLLAR
No. 2 Shaft Trimountain Mine

FIG. 1

FIG. 1A

... the concrete was placed. A foundation ... the ledge by placing heavy steel beam ... the shaft from foot to hanging under the ... the south end plate (Fig. 1A). At the ... was a natural rock ledge or shelf. Starting ... thus formed the steel sets were built up, ... at a time, and concreted in. The work pro-...

... timber on the ends and footwall was taken ... height as was deemed safe; then two or

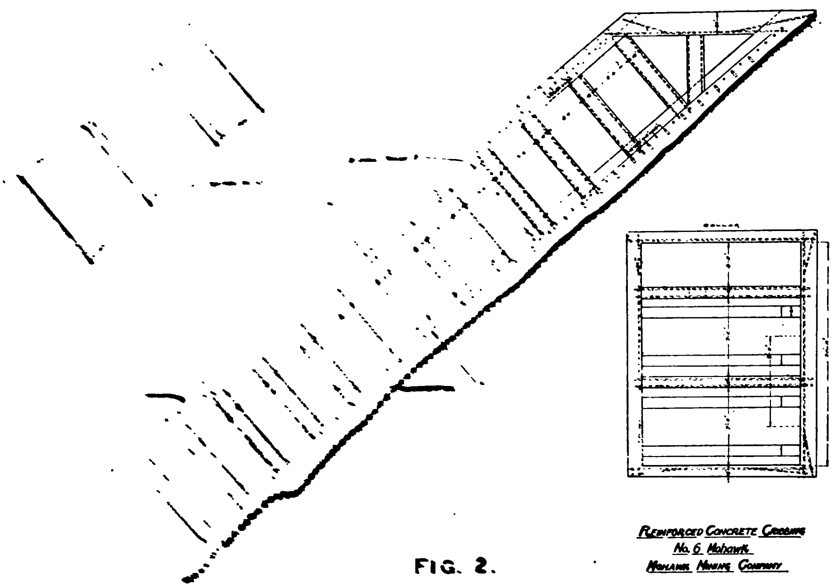
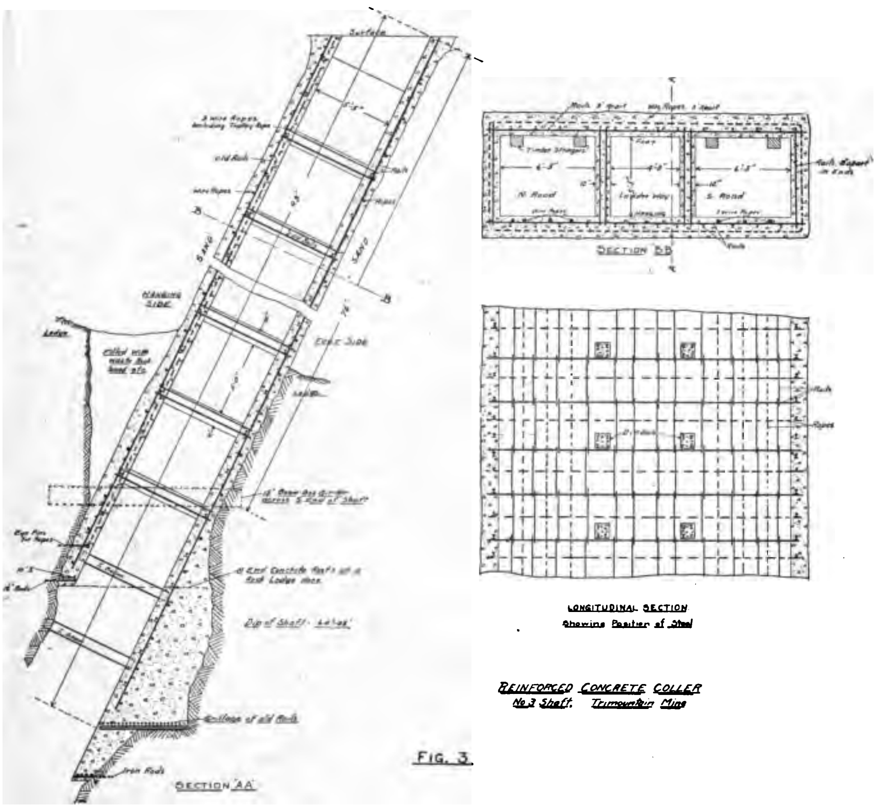


FIG. 2.

... placed and bolted up, after which ... concrete poured. Then an- ... and the operation repeated, ... was reached. Fortunately the ... to be removed, as there ... the new concrete lining be- ... was taken to leave no tim- ... side of the concrete lining ... The sand was care-

fully tamped along the foot-wall as the concrete was finished. One skip road and the ladder way were built first, hoisting going on meanwhile in the other compartment. The skip was then changed over to the completed road, and the other road was built up. The steel sets were 2 feet 4 inches apart in the lower half of the collar, and 3 feet 0 inches apart in the upper half, center to center. The concrete between the sets was re-

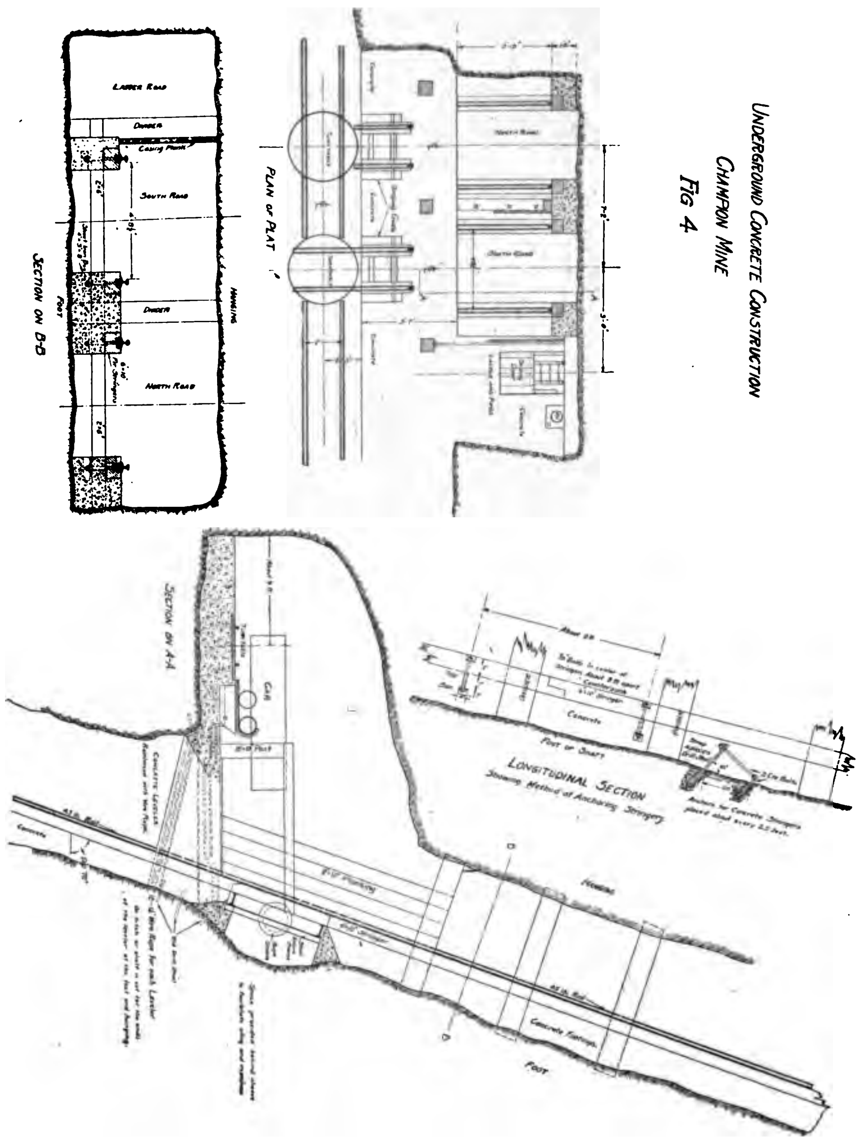


REINFORCED CONCRETE COLLER
No. 3 Shaft. Trimountain Mine

FIG. 3

inforced with ¾ inch rods, as shown by Fig. 1, which also shows the construction of the steel sets and the position of the concrete.

The materials used for the concrete were: Portland cement, coarse amygdaloid stamp sand and crushed trap rock. They were mixed by hand in the proportion 1:3:5,



Underground Concrete Construction
Champion Mine
Fig 4

in the shaft house just back of the shaft and lowered by means of a bucket and trolley, the trolley rope being concreted in on the hanging side as the work progressed. As no difficulty was experienced at No. 2 shaft with the sand running

in, or otherwise, it was decided to build the Nos. 3 and 4 collars of reinforced concrete only, leaving out the steel sets. Fig. 3 shows the construction of the No. 3 collar, which was started in June, 1910, and finished in August, 1910. The materials for the concrete were the same and the work was carried on in the same manner as at No. 2 except that there were no steel sets. The collar at No. 4 shaft was similar to the one at No. 3, except that the dividers were made 12x48 inches instead of 12x12 inches. The overburden at No. 4 shaft was 128 feet deep on the pitch of the shaft, (71°) that at Nos. 3 and 2 being 60 and 80 feet, respectively; but in order to secure a suitable foundation, the No. 3 and No. 4 collars were started some distance below the ledge in the solid rock. The length of No. 3 collar was 93 feet, and of No. 4 was 158 feet.

COMPARATIVE STATEMENT OF COST OF CONCRETE SHAFT COLLARS.

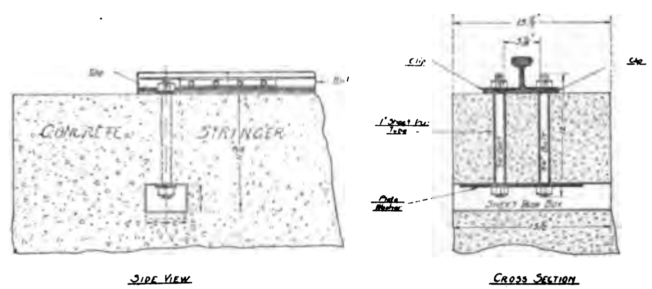
Labor—	No. 2 Shaft.	No. 3. Shaft.	No. 4 Shaft.
Length, to foundation	80 ft.	93 ft.	158 ft.
Shaftmen	$2,019.10	$1,028.85	$1,994.70
Masons	528.51		
Surface labor	301.80	295.50	192.45
Blacksmith labor	360.41	67.55	40.50
Machinist labor	311.76	41.82	27.85
Carpenter labor	144.97	42.73	54.69
Electrician labor	10.84	8.82	8.96
Teaming labor	120.56	74.46	56.64
	$3,797.95	$1,559.73	$2,375.79
Supplies—			
Structural steel	$2,180.56		$ 136.00
Cement—1252 sks. No. 2	588.83		
Cement—1238 sks. No. 3		$ 470.80	
Cement—2169 sks. No. 4			810.09
Stamp sand—11 cars No. 2	159.50		
Stamp sand—3¼ cars No. 3		45.70	
Stamp sand—8½ cars No. 4			123.25
Fine rock, 6 cars	90.00		
Sundry supplies	261.75	102.55	75.91
Freight	215.33		
	$3,495.97	$ 619.05	$1,145.25
Total cost of shaft collars	$7,293.92	$2,178.78	$3,521.04

No. 2 Shaft Collar commenced February, 1907, completed August 1907.

No. 3 Shaft Collar commenced June, 1910, completed August, 1910.
No. 4 Shaft Collar commenced March, 1911, completed August, 1911

Cost per Foot—	Labor.	Supplies.	Total.
No. 2 shaft	$47.47	$43.70	$91.17
No. 3 shaft	16.77	16.66	33.43
No. 4 shaft	15.04	7.25	22.29

Fig. 2 illustrates a reinforced concrete collar designed by Mr. W. F. Hartman for No. 6 shaft, Mohawk Mine, where the dip is very flat (about 38°). The reinforcement was rods and wire rope. The collar was built in 17 days and the total cost was $3.931.00. The length of the collar was 100 feet. A pit was first excavated at the shaft site. Then the forms



Details for Concrete Stringers
Showing Old Method of Fastening Rails.

FIG. 5.

were started at the bottom and built up as the work progressed. The concrete was mixed on surface and run down to the working platform in an iron trough. The use of concrete for plat floors, levelers, stringers and dividers is becoming quite common.

Fig. 4 shows a station or plat in one of the Champion Copper Company's shafts, and indicates the manner in which the levelers are reinforced. This illustration also shows the method used for concrete stringers. At first an all concrete stringer was built after the manner in use at the Ahmeek

Mine as designed by Mr. W. J. Uren, to which the rail was bolted by means of bolts and clips as shown in Fig. 5, but because of the hard rigid roadbed thus formed the wear and tear on skip and rails was very great, and the bolts and clips were continually working loose. The scheme was therefore abandoned in favor of a combination wood and concrete stringer.

Fig. 4 shows the method in use at the Copper Range Con-

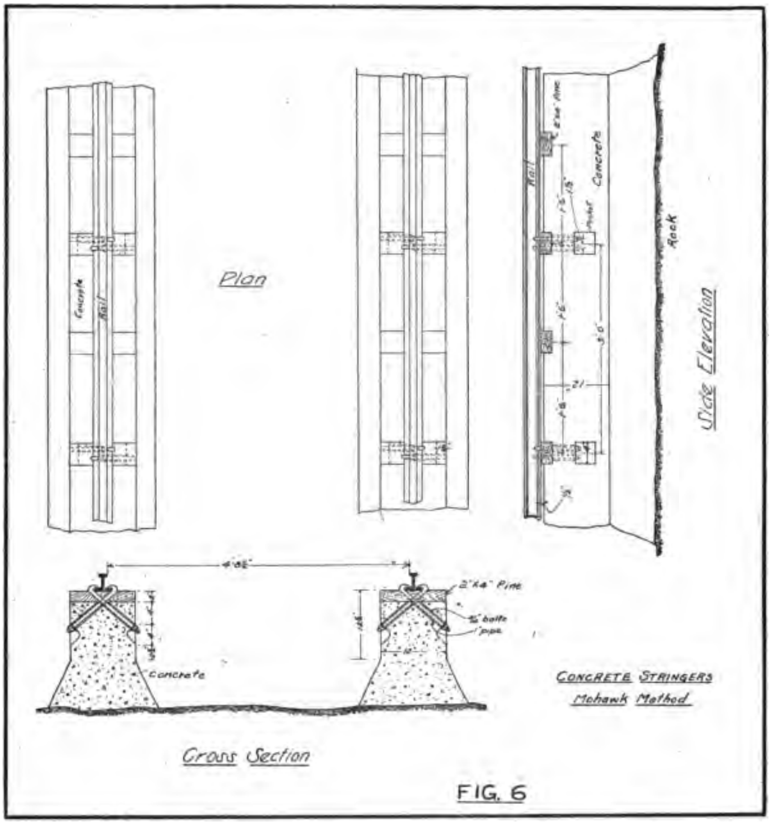


FIG. 6

solidated Company's mines, and the Mohawk and Wolverine scheme is illustrated in Fig. 6. Both methods made a very satisfactory road bed.

At some of the mines where the foot is subject to "heaving" concrete stringers cannot be used advantageously.

In sinking through some loose ground at one of the Champion shafts it became necessary to close-timber, or line the shaft. Concrete 12 to 18 inches thick was put in, reinforced with old rails and wire rope. The concrete extended across the hanging and down on both ends, and sometimes across the foot, and there were also heavy concrete dividers 4 feet high by 10 inches thick, placed 10 or 12 feet apart. At several levels the whole plat was arched over with reinforced concrete. This lining has been in place about two years and has proven satisfactory.

Drift sets built of concrete have been tried to some extent at the Wolverine and Mohawk mines in some of their cross-cuts where loose ground was encountered. These sets consisted of legs 6x6 inches in section, and a cap 6x8 inches, reinforced with ½ inch rods and wire rope. Concrete planks, reinforced with Kahn expanded metal, or woven wire, were used for lagging. Above the caps they were 4x14 inches in section and behind the legs 2½x14 inches.

The use of reinforced concrete in the form of shaft sets and lagging is well described in a paper read before the Michigan College of Mines Club, at Houghton, Mich., by Mr. E. R. Jones, who has kindly given his permission for the use of the following excerpt:

"For a number of years solid concrete and reinforced concrete shaft collars and shafts have been in vogue where the conditions warranted a shaft of any degree of permanence, but not until nineteen hundred and nine was reinforced concrete tried as a substitute to take the form and similar methods of installation as the long-used timber sets for shaft purposes; namely, at the Nos. 3 and 4 shafts of the Ahmeek Mining Company.

In the beginning, two distinct kinds of material were used; a good grade of gravel and natural sand from a local pit; and the trap rock, through which the shafts were sinking, together with clean conglomerate sand from the Calumet & Hecla mill. Sets were moulded from these two classes of material and installed with equal partiality and subsequent service has proven both to be equal to the demands made upon them. Pieces set aside for the purpose were allowed to season sufficiently that they might be given a fair competitive test, and it was found, on comparing the fractures in the two combinations of material, that the sand and cement filling the

spaces between the rounded pebbles broke away from them, while the fracture in the trap-conglomerate same combination continued through the larger elements of the mixture. The gravel mixture could doubtless have been improved considerably by careful washing, but the cost of preparation, compared with the trap rock and conglomerate sand, prohibited its use in this particular case.

The materials finally used were as follows:

No. 1 Portland Cement. Conglomerate sand. Trap rock trommeled over ¾-inch through screens. The proportions used were 1:3:5 in wall plates, end plates, and dividings, and 1:2:4 in studdles. The reinforcement in wall and end plates consisted of three ¾-inch Monolith steel bars with ¼-inch webs, crimped onto them, together with two straight ¾-inch Monolith bars. The dividings were reinforced by four ½-inch Monolith steel bars wound spirally with ¼-inch steel wire, the whole presenting a column with square cross-section. Studdles were reinforced with two pieces of old wire rope 1¼ inch in diameter. Reinforced concrete slabs were moulded for the shaft lining, the material used being fines of trap rock under ¾-inch, conglomerate sand and Kahn expanded metal as reinforcement. The mixture used for slabs was 1:2:4. By way of experiment, the writer selected a piece of No. 1 hemlock plank of the same length, width and thickness of a concrete slab, which had seasoned for one year, supported them at either end, and placed them side by side, and then applied an equal pressure across the center of each. Three failure cracks appeared in the concrete slab just previous to the breaking of the hemlock plank, although total collapse of the concrete slab did not occur until the pressure was considerably increased. While the method of the test employed was crude, it proved to the satisfaction of the writer that the concrete slab was much superior in strength. Considering the rapid decay of timber used as shaft lining no further comparison of the two is necessary.

In the moulding of the concrete sets 2-inch No. 1 white pine was used in the construction of the forms. These were soaked in Delaney's wood preservative, and repainted with preservative on the interior each time before setting up, thus insuring them against warping and prolonging their lives indefinitely, as well as securing a smooth and easy parting from the concrete when removed. A Smith barrel type mixer was employed in preparing the charge for the forms. The amount of water used in the mix was such that, when the batch was piled, it settled rapidly without agitation. A dryer mix was attempted by way of experiment, but due to the amount of reinforcement employed, it was found impossible to ram the dryer mix into place.

The labor involved in making consisted of two carpenters, setting up forms and keeping them in repair; one man wheeling forms onto skidways ready for filling, returning used forms to shop and cleaning the same; one man feeding mixer from stock piles of rock,

sand and cement; one man delivering mix to forms and shoveling material into place; and one mason ramming charge into final position. With this combination of men as many as four complete sets, consisting of 64 separate pieces, have been moulded in one day of nine hours. In ordinary weather, the sides of the forms were allowed to remain in position over night, and then removed, while the bottoms were left in place another twenty-four hours. The bottoms were removed by turning the pieces on their sides, where they were left to harden one day longer before removal to the stock pile. All through the process of removal the sets were handled with the greatest care in order to preserve the appearance of the set and prevent cracking, which might not develop to the eye until weathered. All skidways used in making and storing were brought to a level to prevent warping and bending while the sets were green, to insure a perfect fit underground, for unlike timber, the concrete set cannot be brought to place unless perfectly true. Sets should not have been used under sixty days after removing forms, although we, through the reduction of the stock piles, have been forced to install pieces of fourteen days set, but the greatest care was observed in handling and putting in place underground. Concrete sets one year old, which have been subjected to all manner of weather, can be abused somewhat and handled almost as carelessly as timber.

As before stated, the above mentioned sets were made for the Nos. 3 and 4 shafts of the Ahmeek Mining Company. The shafts are of the three-compartment variety—two skipways and one manway, dipping at an angle of 80 degrees. The outside dimensions of the compartments are:

Skipways—7 feet 6 inches high, 6 feet 10 inches wide.

Manway—7 feet 6 inches high, 3 feet 0 inches wide, with the end plates and dividings, making the greatest span of 7 feet 6 inches. Offsets were moulded in all plates 5 inches from the inside face to accommodate lining slabs. Also, holes were cored for the use of hanging bolts and bracket bolts. The wall plates, end plates and studdles have a cross-section of 80 square inches, dividings 81 square inches. The percentages of reinforcement are approximately as follows·

Wall and end plates	5 per cent
Dividings	5 per cent
Studdles	3 per cent

It was found advisable from the beginning, because of the great weight of the wall plates, to mould them in two sections, one section spanning the ladder way and one skipway, and the other section spanning the remaining skip compartment. These two sections were connected when in place by two bolts passing through holes, cored for the purpose, and two straps of iron spanning the splice. Studdles were made for 4 feet 0 inches, 5 feet 0 inches, and 6 feet 0 inches sets to accommodate the ground passed through.

The weights of the different pieces comprising the set are as follows:

	Pounds.
Long section of wall plate	1,035
Short section of wall plate	700
End plate	600
Divider	645
Three feet 3-inch studdles	268
Complete set of 16 pieces	8,104

Taking the weight of No. 1 Western fir, which has been exposed to the weather in stock piles, as 33 pounds per cubic feet, the above concrete set weighs almost three times that of a 12x12-inch timber set which the concrete set is intended to replace. Because of this additional weight of the concrete set, it was found necessary to increase the usual five or six men on the timber gang to seven in number. In a vertical shaft, to which the concrete sets are especially adapted, the number of men per gang might again be reduced. The sets are hung or built as the ordinary timber sets, only requiring an additional rope and block to swing the pieces in place. After the sets are wedged to line, bottoms are put in between the plates and the surrounding shaft wall, and the set is then tied to the shaft wall by means of concrete, in the proportion of 1:3:5. The concrete slabs are then put in place, and loose rock thrown behind them, filling up what space still remains between the set and the wall of the shaft.

After the set is in place, it is extremely important that it is well protected from the blast, for, unlike the timber set, concrete will not stand the blast. For the purpose, the writer used flat timber and steel plates chained to the under side of the plates and dividings, and even this precaution was at times inadequate. Where the ground was breaking easily, the sets have been as near as twelve feet to the miners, and again when the ground was especially refractory, sets forty feet from the blast have been cut out. It is obvious that it is well to keep as far behind the mining as the ground will permit. In dangerous ground, which required timbering close up to the sinking, timber sets were used, but, had not time played an important part in the sinking, no ground was met in which concrete sets could not have been installed. With a gang of seven men, one complete set can be installed in a nine-hour shift. This permits a sinking rate of better than one hundred feet per month, which was accomplished at the Nos. 3 and 4 shafts.

The comparative cost of the concrete set and timber set, delivered at the shaft collar, is striking. The concrete set was delivered for $22.50, the timber set for $37.60. These figures are based on:

Western fir at $28.00 per M., f. o. b. car.

Crushed rock at 35c per yard, f. o. b. shaft.

Conglomerate sand at 60c per yard, f. o. b. shaft.
No. 1 Portland cement at $1.15 per bbl., f. o. b. works.
Reinforcement at $12.00 per set, f. o. b. factory.

The Ahmeek Mining Company, I believe, was the first to adopt the concrete stringers, and the Mohawk Mining Company soon followed with their use. At the Ahmeek, these stringers have been in continuous use since the beginning of operations and have required no repairs. Superintendent Smith of the Mohawk has informed me that soon after the stringers were installed, skip repairs increased about one hundred per cent. The stringer being entirely rigid and the skip also of rigid construction, the axles of skips were found to be crystallized and the rivets working loose. This feature was overcome by moulding 2-inch pine strips, after preserving them in Delaney's Wood Preservative to prevent decay, into the stringers at intervals of three feet, allowing them to project one-half inch above the face of the stringer, and resting the rail thereon. The pine strips have been in place four years, and none have been replaced to date, and skip repairs have been reduced to normal. Possibly because of a differently constructed skip, Ahmeek repairs were not abnormally high but the same racking of the skip body occurred and the Ahmeek Company has adopted the Mohawk feature and expects to profit accordingly.

Concrete plats, or stations, have been in use at both the Ahmeek and the Mohawk for some time. They differ from the timber plat in outward design only in the cross-section of the members, which are 9x12 inches, and are reinforced with old rail and wire rope, and replace the 12x12 inch and 12x14 inch timber formerly used. Holes are cored to accommodate gates for skip and dump doors and tram rails are imbedded in the concrete, making the use of spikes unnecessary. When turn-tables are used on the back of the plat, the rigidness furnished by the concrete insures the trammers against derailed cars, resulting from a tilted table.

At the present time, our company is installing reinforced concrete dividings to replace the practice of putting in 10-inch flat timber. In cross-section they are 9x12 inches, and are reinforced by old rail. On the ladder road, they are placed six feet from center to center and between the skip compartments are put in as often as the hanging requires. Since the casing along the ladder road performs no other office than the protection of the men while on the ladder, or in case of a fall, plank is used for the purpose, and a 3-inch hemlock strip is moulded into the dividings to facilitate the fastening of this casing."

Concrete floors for shaft houses are being tried at several places and are proving satisfactory in many respects, although subjected to the hardest usage. The floors built at the Champion Copper Company's shaft houses were made 6 inches thick

of 1:3:5 concrete, with a top finish 1 inch thick of 1 to 2 Portland cement and coarse stamp sand. The total cost was 13 cents per square foot. The materials used for the concrete were crushed trap rock, ccarse stamp sand and Portland cement.

Question has been raised from time to time as to the suitableness of wire rope for reinforcement. Some tests of concrete beams reinforced with wire rope strands were made at the Baltic Mine in 1910 by Mr. C. G. Mason and the results are given in a paper written by him and published herein with his permissison.

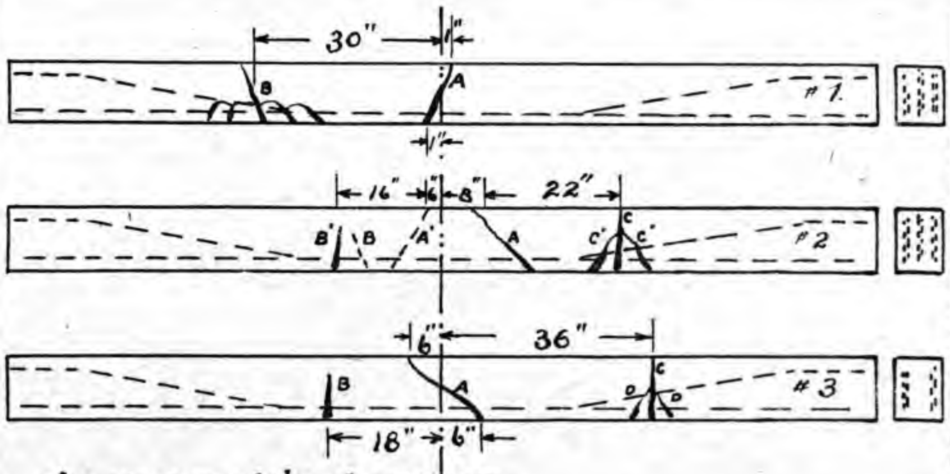
EXPERIMENTS MADE AT THE BALTIC MINE OF THE COPPER RANGE CONSOLIDATED COMPANY ON "CONCRETE BEAMS" REINFORCED WITH WIRE ROPE, C. G. MASON, E. M.

Through the General Manager's (Mr. Denton) consent, I was enabled to present results which might be of interest to those who are in doubt as to its practical use. You who are familiar with the use of wire rope in mining, know the number of discarded "reels" in the scrap heap, which are difficult to dispose of, being of apparently no commercial value. This rope is 1¼ inches in diameter, consisting of six strands wound around a hemp center; each strand having 19 wires. First, we made some tests on ropes which had been used from one to two years and found the ropes to contain an average area of .63 inches and would stand a breaking load of 45 tons to the original guaranteed load of 69 tons. We then unwound these ropes and tested several strands, each averaging an area of .1052 square inch. It took a load of 4,000 to 9,000 pounds to straighten the twist, with an elongation of 1.40 per cent. With a gradually increasing load the strands broke, having an ultimate load of 12,000 to 23,000 pounds. The surface of all wires available for bonding surface was found to average 1.55 square inches. See Plate 1, Tables 1 and 2. A ⅝ inch round steel bar is the nearest equivalent, having 1.18 square inch of surface. We have not, as yet, been able to test out these strands in concrete blocks to determine the bond.

Previous to these experiments, three concrete beams reinforced with these strands were made and tested. The ingredients used in this concrete were: Alpha cement, coarse stamp sand (or tailings) and ungraded small trap rock from the mine. The average weight of this reinforced concrete was 150 pounds per cubic foot. For data regarding beams and method of failure see table 3, plate 1.

Beam No. 1. This beam was loaded with a gradually increasing load and showed the first crack at point A, when the load was 7,480 pounds and the deflection was .20 of an inch. The crack at B appeared

Plate I



Average area of strands = 1052 sq. in.
Average diam of strands = .041 sq. in.

3.

$\frac{M}{bh^2}$	No. of Beam	Prop. of concrete	Age in days	Span in feet	No. of strands	= in size to bars	Diam of str'nds	% of steel	Depth of Beam	Dist of steel from top	Breadth of Beam	Max load #s centre	at 1st crack Load	at 1st crack Def.	ult. Defl't'n
616	1	1-2-4	98	10	6	3/8	0.41	0.93	10"	8½"	8	11880	7480	0.20"	0.50"
750	2	1-2-4	100	10	6	3/8	0.41	0.93	10"	8½	8	14480	6480	0.20	0.82
542	3	1-3-5	100	10	4	3/8	0.41	0.62	10"	8½	8	10480	6480	0.36	0.56

NO 1.

	Diam of Single wire	Circum of Single wire	Area of wire one strand
1	.067	.2105	.0035
2	.068	.2136	.0036
3	.071	.2231	.0040
4	.068	.2136	.0036
5	.094	.2953	.0069
6	.069	.2168	.0037
7	.068	.2136	.0036
8	.097	.3047	.0074
9	.095	.2983	.0071
10	.082	.2576	.0053
11	.082	.2576	.0053
12	.096	.3016	.0072
13	.082	.2576	.0053
14	.096	.3016	.0072
15	.096	.3016	.0072
16	.084	.2639	.0055
17	.081	.2545	.0052
18	.082	.2576	.0053
19	.085	.2670	.0057
			.1026

NO 2

1	.069	.2168	.0037
2	.095	.2983	.0071
3	.070	.2199	.0038
4	.095	.2985	.0071
5	.069	.2168	.0037
6	.093	.2935	.0069
7	.070	.2199	.0038
8	.095	.2983	.0071
9	.072	.2261	.0041
10	.095	.2983	.0071
11	.070	.2199	.0038
12	.094	.2953	.0069
13	.085	.2670	.0057
14	.084	.2639	.0055
15	.084	.2639	.0055
16	.084	.2639	.0055
17	.091	.2859	.0065
18	.084	.2639	.0055
19	.084	.2639	.0055
			.1078

soon after the one at A. Just before reaching 10,480 pounds the other cracks in the vicinity of B began to appear; with this load the deflection was .30 of an inch. The greatest load carried was 11,880 pounds, with a deflection of .50 of an inch. The ultimate failure occurring at B.

Beam No. 2. In this beam with a load of 6,480 pounds two cracks (A and A) appeared on opposite sides and the deflection was .20 of an inch. The remaining cracks seemed to appear at the same time, at a load of 10,480 pounds, with a deflection of .32 of an inch. All cracks gradually extended towards the top of the beam, with a load of 12,480 pounds, and a deflection of .46 of an inch. Finally with an ultimate load of 14,480 pounds and a deflection of .82 of an inch the beam failed at C.

Beam No. 3. Here, with a gradually increasing load at 6,480 pounds and a deflection of .36 of an inch cracks at A appeared. Cracks B, C and D appeared in order when the load was 10,480 pounds, and the deflection was .56 of an inch. This was the greatest load; ultimate failures occurring at D. In these beams all the cracks as they extended towards the top, gradually widened at the bottom. The failure of these beams was not caused by the slipping of the strands; which proved a very satisfactory result in this test. Had they slipped, the grooves formed by the strands, would have crushed or sheared the ridges of the concrete, but nothing of this nature occurred.

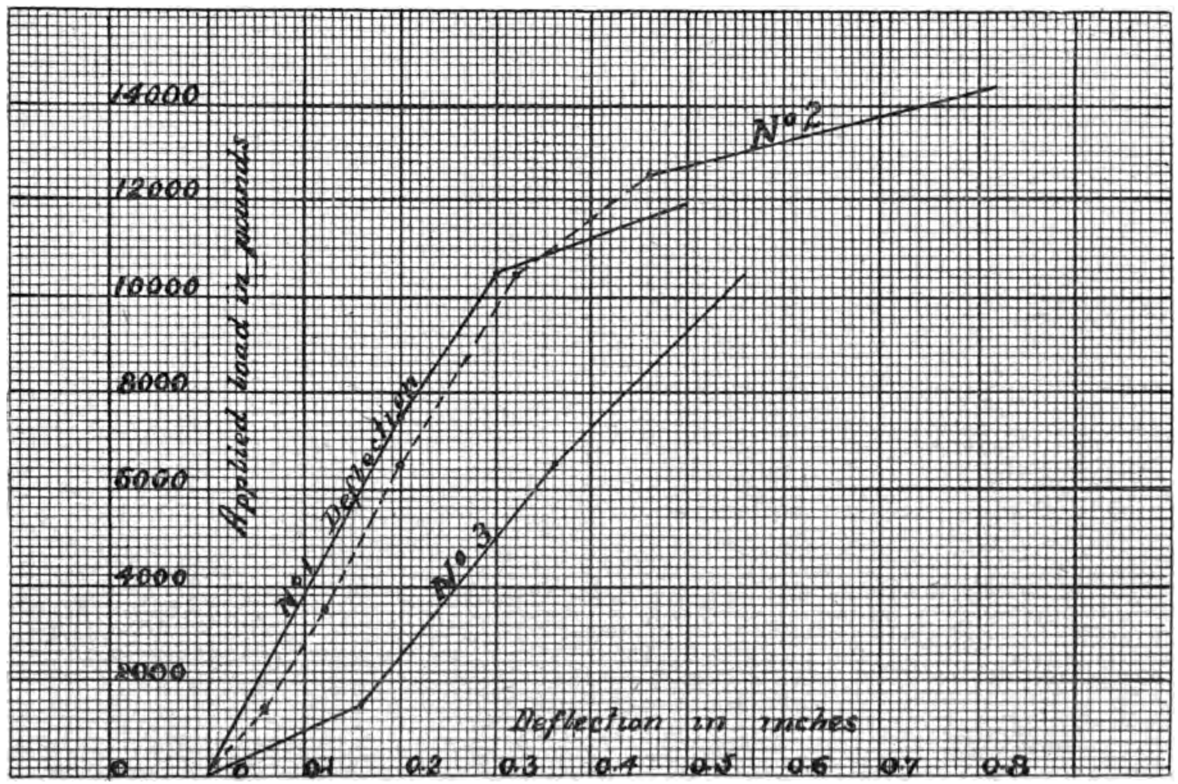
Plate 2 shows the deflection curves for the three beams.

As a result of these tests we were able to compute by the straight line formula and the usual safe stress of 650 pounds per square inch in the concrete, and the per cent .93 of steel, the safe load at the middle of a beam of this size and span, which is 2,180 pounds. The average ultimate breaking load of beams No. 1 and No. 2 is 13,180 pounds. This would indicate a factor of safety of $\frac{13,180}{2,180}=6$

Apparently then, it ought to be perfectly safe to use these strands with the per cent of steel about ¾ to 1 per cent. In closing, I would suggest, when the area of steel is large, if possible, to use the whole rope, and where the area is small to use the strands in pairs as unwound from the rope. Since they can be placed in almost any position, requiring less labor and time than the placing of single strands.

Concrete seems well adapted to underground use, especially as illustrated here. The effect upon it when the surrounding ground begins to give under pressure, or is under great stress, is still to be learned by experience. The Copper Range Consolidated Company is inclined to confine the use of concrete in shafts at depth to stringers for the road bed and to the sollars and roof at stations, with the idea that should

Plate 2.



Deflection of reinforced concrete beams.

the ground around the shafts fail it will be easier to maintain the shaft if timbered than if lined with concrete. On this point, however, it is in doubt.

DISCUSSION.

MR. DENTON: This involves descriptions of the more important applications of concrete at various mines of this district. On the excursion yesterday you saw one example, referred to in this paper, of concrete sets at the Ahmeek. (Illustration is shown elsewhere). Mr. Mercer is present and will be glad to answer questions or give further explanations of anything pertaining to the paper.

MR. HEARDING: Regarding the cost of rock and sand, as given by Mr. Jones for the Ahmeek, the figures are unusually low and could only be used in this district where an immense amount of crushed rock is available. I doubt if any of the figures could be reproduced in any other part of the country.

MR. DENTON: On the subject of the cost of concrete, as Mr. Hearding says, this district is, of course, very fortunate in having a plentiful supply of the base materials, that is, in the rock and sand. Furthermore, for the underground concrete, we do not have to introduce crushed rock from the surface, as the finer material taken from the shafts and drifts in barren ground is more than sufficient to do all of the ordinary concrete work. Occasionally, on specially heavy jobs, rock has to be sent down from the surface, and in that case the finer material from the nearest waste pile is used. The sand is obtained by gravity from the discharge launders at the stamp mills and the main expense there is the haul back which, being done by the empty rock cars, is not a serious expense. Mr. Jones' figures are I think correct and definite and we estimate that our concrete work underground is cheaper than if the same space or area were covered by timber, using fir timber. We have not kept the figures accurately enough to give them definitely, but we have kept track of the cost, for instance, of the concrete work in one lift of a shaft and then attempted to estimate what the same work would

have cost on the old basis, using timber. It is always a little uncertain how much timber would have been used if the job had been done by timber, and so the figures are not as definite as one might wish, but we estimate that there is a very considerable margin in favor of concrete.

MR. HEARDING: My reason for speaking of the cost was that I was very much surprised to find at what cost crushed rock and sand could be produced at the mine. I doubt if at any mine in the iron country it can be produced for less than $2 a yard for both, but at a cost of 35 and 60 cents, it would make the advisability of concrete in place of timber very marked.

MR. DENTON: Those prices are rather high; we can introduce good crushed rock into this country from below for about, I believe, $1 per yard; the cost of sand depends on the locality, but there are few localities in which sand of a sufficiently good quality for concrete work cannot be found, although it varies. Then there are many gravel beds and a good gravel bed is as satisfactory for that sort of work as anything else and perhaps a little cheaper to handle, requiring less sand. Referring further to the matter of expense, Mr. Mercer's paper gives in full detail the exact cost of concrete collars and shows the improvement coming from experience, as at the Trimountain mine three were put in. The first involved a great deal of reinforcing and the use of steel sets and the steel sets were a very expensive part of this structure, and the last one involves only reinforcement with waste material, wire rope, etc. These figures being divided into the various items of labor and supplies and the supplies further subdivided, enable one to estimate quite closely the approximate unit cost for such work.

CONSTRUCTION OF INTAKES AT THE MILLS OF THE TRIMOUNTAIN AND CHAMPION MINING COMPANIES.

BY EDWARD KOEPEL, BEACON HILL, MICH.

In 1909 the Baltic, Trimountain and Champion Companies, now subsidiaries of the Copper Range Consolidated Company, found it necessary to procure 20,000,000 gallons of water per day for each of the three mills they had planned to erect on the shore of Lake Superior. The Baltic, with the Atlantic Mining Company, impounded the water of the Salmon Trout river with a steel gravity dam and thereby secured an ideal water supply. The Trimountain and Champion were forced to draw upon Lake Superior for their supply. On the west line of Township 55 N, Range 35 W, they found two mill sites where one intake might serve both. Soundings disclosed a sandstone bottom, ideal for anchoring a crib and pipe. It was free from sand or boulders and inclined about 20 inches per hundred feet.

A timber crib was designed, 56 feet by 42 feet, with two wells and two intake pipes, one for each mill. Shortly afterwards Champion selected a site three-fourths of a mile further west and planned to secure water through a tunnel. For several reasons Trimountain decided to stick to the pipe and intake crib, the sandstone bottom was criss-crossed with fissures 1½ to 8 inches wide, with a soft white filling into some of which we drove a steel bar six feet. It was possible that a tunnel might open enough fissures, making it necessary to put in an air lock and work under air pressure or to abandon the tunnel entirely, and put in a pipe.

Another reason was the need of time. It was expected

one-half the mill might be finished in a year. A tunnel 1,400 feet long, shore shaft and lake intake might take two years to complete, with a possibility of the intake being further delayed by winter. We believed a crib and 1,400 feet intake pipe, including a trench on the shore end 400 feet long, could be finished in four months, the costs of which should not exceed those of a tunnel. We also believed the pipe could be extended when the tailings encroached upon the inlet.

Trimountain started this work in June 1910, blasting the trench on company account and contracting for the pipe and crib The contractors did well but the trenching suffered not only from storms, which were expected, the shore being exposed from all sides but the south, but from inability to clean out the trench after blasting. The drill platform for this trench was suspended from two wire cables fastened to timber supports in the lake. At each setting of the platform three holes were drilled on each side, pointing toward each other, and two vertical on the center line. This covered 12 feet of the trench. Single-bit jump drills were used inside a 2½ inch pipe, three men to a drill. When the drill was withdrawn the hole was charged through the same pipe and the battery firing connection was made. The spuds or legs of the platform were then raised, the position on the cable was marked and the platform was hauled shoreward before the round was blasted.

Two settings a day were drilled, covering 24 feet of the trench, and the entire length was blasted before we began clearing out. For cleaning away the blasted rock a five-pronged scraper made of rails was rigged to the wire cable that carried the drilling platform, and was manipulated with an engine. The blasting had looked well done but we could remove only three feet of broken rock from the trench. It was redrilled many times, but only a little rock could be removed after each blasting.

The crib was placed and, by the aid of a diver, 700 feet of pipe laid and partly secured by the middle of September. Another diver was engaged to anchor the pipe already laid.

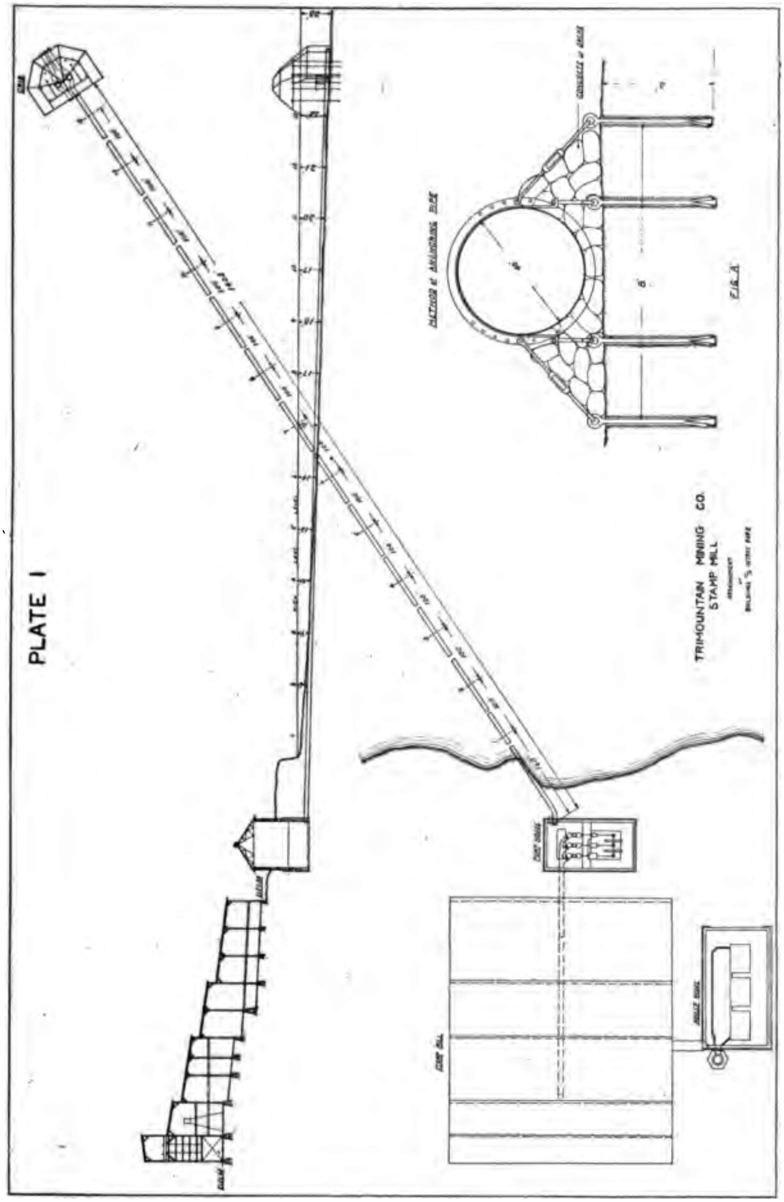


PLATE I
TRIMOUNTAIN MINING CO.
STAMP MILL

It was then thought possible to finish before winter. On September 17 a violent storm wrecked the 10 by 10-inch legs of our wire cable support and the cables disappeared in the water. After the weather moderated, of the 700 feet of pipe laid from the crib toward the shore, the divers found only 300 feet in place. This was then securely fastened and the job was stopped for the winter.

The following May work was resumed with two divers and two crews. It was evident the job was bigger than at first anticipated. The delivery of mill machinery was much delayed, but it still looked as if the mill would be ready before the intake. The damage by the storm taught us to secure the pipe better, and we bedded it in for half its circumference with concrete put in place in burlap bags.

In September the pipe was connected with the shore end behind a coffer dam. We then learned why the trench was so difficult to clean out. The crevices caused by the blasting were filled with a clayey cement, the force of the blast turning some of the sandstone into clay, this clay cementing the broken rock in place. The sandstone underwent the same change when exposed to the weather, turning into a clay that made a good cement.

Early in November, 1902, the mill commenced operations. The intake included 1,400 feet of rivetted pipe made of ¼-inch steel plate, in sections each 25 feet long connected with steel flanges formed from ⅜-inch steel plate. On the shore end 400 feet lying in a trench was covered with sand and broken stone. The rest lay on the natural rock bottom. For anchorage 240 holes were drilled, two at each flange and two more sets or pairs along each section. These were close to the pipe. Each hole was fitted with a 1¼-inch eye bolt and fastenings were made as shown at A, plate 1. Concrete in sacks was placed the whole length of the pipe, also shown at A, plate 1. The top of the pipe was left bare. The crib in place had cost $11,972.81. The pipe in place had cost $41,005.55.

In the winter of 1902-1903 the ice dented the pipe, and in the spring the exposed part was covered with concrete put in place in sacks. This required 34,452 bags and cost $11,064.74. The intake was then considered finished and had cost as follows:

Labor	$15,546.51
Divers	9,892.50
Consulting engineer	500.00
Supplies	16,750.68
Pipe, 1,900 feet of 40-inch	8,380.60
Crib in place	11,972.81
Total	$64,043.10

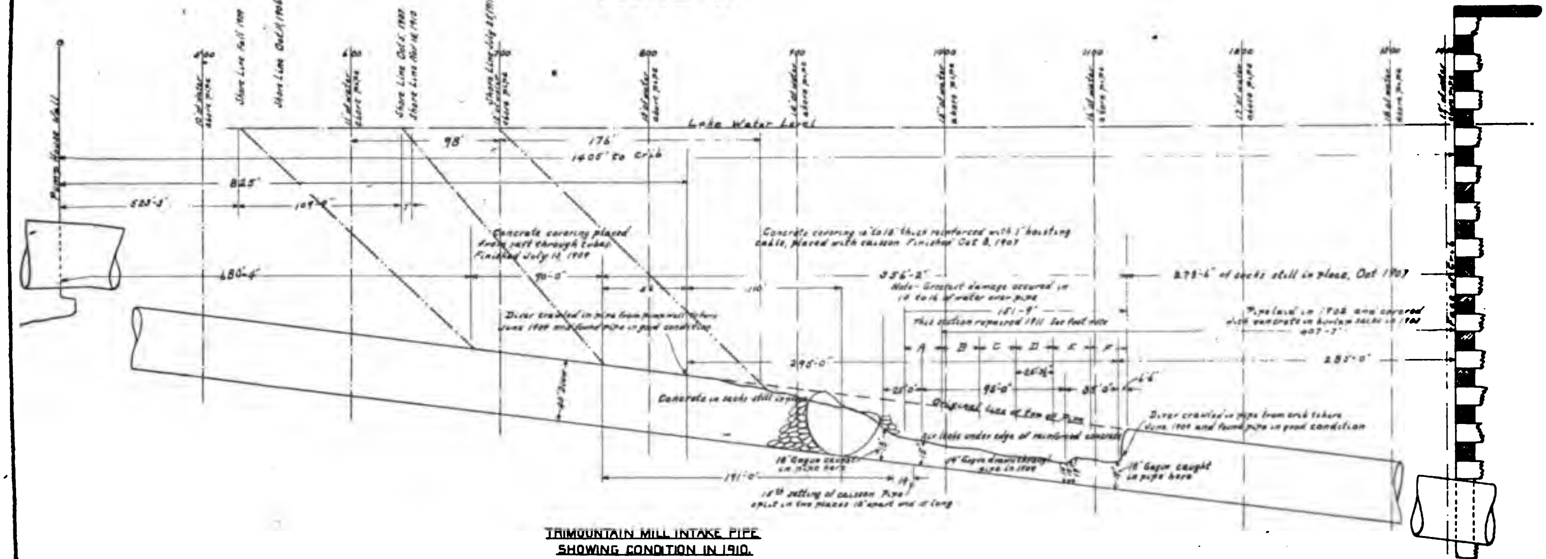
In 1905 it was necessary to repair and reballast the crib. The ballast was gone from the inside, the riprapping from the outside, and the bottom timbers had their corners scoured off. The crib was reballasted with mine rock at a cost of $821.43.

Trouble then began because of tailings from the mill, wood chips and bark finding their way into the well and the pump after storms, cutting out the pump valves and stopping up pipes in the mill. We believed this resulted from leaks in the pipe but an examination disclosed no holes, although the ice had worn the concrete from the pipe for over 200 feet. Because of the murky water all the inspections made by the diver were in almost total darkness and he worked by the sense of touch alone, crawling along both sides of the pipe, feeling with both hands for something wrong.

In 1907 these troubles became so acute that another examination was made. Fortunately the water was clearer and the pipe could be seen from the surface. The diver reported one-third of the covering gone, the pipe flattened down to the ridges of concrete which remained on each side, the rivets in some of the girt seams and in some longitudinal seams sheared, and the pipe split open in three places, one ten feet long and opened three inches.

We covered 356 feet of the damaged pipe with concrete, reinforced with old hoisting cable, using a movable caisson to

PLATE II



WORK DONE ON INTAKE PIPE IN 1911.

Section A pipe removed and replaced by good length and covered with concrete in July. Shoreward from A diver found pipe in good condition. Toward crib diver found section B flattened at its center to about 30 inches but could not crawl beyond center of section C as pipe was flattened to about 20 inches. Pipe held up well at flanges. August 6 removed section E. August 14 removed section D. August 14 to 20 with hydraulic jack shoved up one-half of section F, remainder toward crib in good condition, and all of sections B and C. Gauged these pipes and in no place found them smaller than 34 inches. August 29 put in section of new pipe at D and on September 1 one at E. Between B and E placed 3 inch tapered filling piece, bolted up flanges and covered all six sections with concrete. Concrete lowered from scow in bucket. Finished September 9, 1911.

put it in. The caisson was 18 feet long, 10 feet wide and 8 feet high. The air lock for men was built of a piece of intake pipe. The locks for concrete were 10-inch pipe. The lifting tanks were two sections of intake pipe. The picture (a) shows the construction very well, the picture (b) shows the caisson in place. Ten working days were required to cover 356 feet, but the caisson was in the water from September 15th to October 8th, stormy weather delaying the work. Work was done during the day, using mill labor. We were able to take care of the regular stamp mill work at night. At the long split in the pipe short pieces of railroad steel were laid across the pipe, burlap was put over the split and the whole concreted over. The mill was shut down for six hours after the concrete was placed there. The caisson work, costing $5,-988.97, went smoothly but results were indifferent for reasons noted later. Two Rand three-drill compressors furnished air, and fire hose conducted it to the caisson.

Most of the water pumped must have been sucked in through this big split and the pipe beyond it nearly filled with sand, because on starting the pump six hours after finishing the repair to the split the water in the pump well was lowered very much for a little while and then came back to its normal level. Two years later we pulled a hose out in the pipe from the shore end and pumped in some air, thus locating the leak by the bubbles. It was then learned that our cementing job on the split had been partly spoiled that night, the cement having been sucked out of the concrete for six inches above and clear out to the side, leaving the top in good shape. Consequently the sand leaked in the same as before.

In 1908 it was found the ballast was again out of the crib and some of the timbers on three walls were gone, on one wall the gap being 14 feet high. These gaps were closed by a diver with 2-inch plank. The crib was ballasted with 200 yards of concrete poured into the compartments through tubes in the roof. The tubes were kept full and withdrawn as the pile came up. This concrete work cost $1,411.01, or about

$7.00 per yard. The rising and falling of the water in the crib washed most of the cement out of the mixture. Although the concrete on top got fairly hard the water could be seen sluicing back and forth through it. The next year the crib was again empty.

In 1909 it was hard to get enough water through the pipe. The diver crawled in from the crib end for 285 feet and found the pipe flattened. He then crawled in from the shore end 825 feet, the limit of his air hose, and found the pipe in good shape. A parachute, attached to a suspended cord, was sucked through the pipe while the pump was in operation. With the suspender cord a ¾-inch rope was pulled through. With the rope a skeleton torpedo-shaped frame, 18 inches in diameter, was pulled through from the crib with another rope attached to pull it back. At 291 feet the frame stuck a little but was pulled 35 feet further where it stuck. It was withdrawn and a 15-inch frame substituted, but could go only a little further. Then a 14-inch frame was tried. It was tight about the same place the 15-inch frame stuck, but it was pulled through into the pump well. The 18-inch frame was now pulled out from the shore end until it stuck, showing only 155 feet of the total 1,400 feet was damaged, and that all the damage, from the ice, occurred in 14 to 16 feet of water. With this information the tracing shown on Plate II was made.

In 1911 we replaced the 155 feet of damaged pipe with good pipe. To remove the concrete placed with the caisson and reinforced with hoisting cable the diver blasted the concrete until some of the cable was exposed. A hook was inserted, with a strong wire rope attached to a powerful winch, which removed the large pieces. The smaller ones were hoisted in a bucket. The pipe was cut in two by blasting with 88 per cent dynamite, and a section removed. The accumulated sand was loosened with a hose and the pump sucked it into the well. A new length of pipe was put in and covered with concrete lowered into place with a self-dumping

bucket holding 8 cubic feet. Two other sections were then removed and the rest of the dented pipe was sprung back nearly to its original diameter with a 90-ton hydraulic jack actuated from the scow above. Two new sections were then

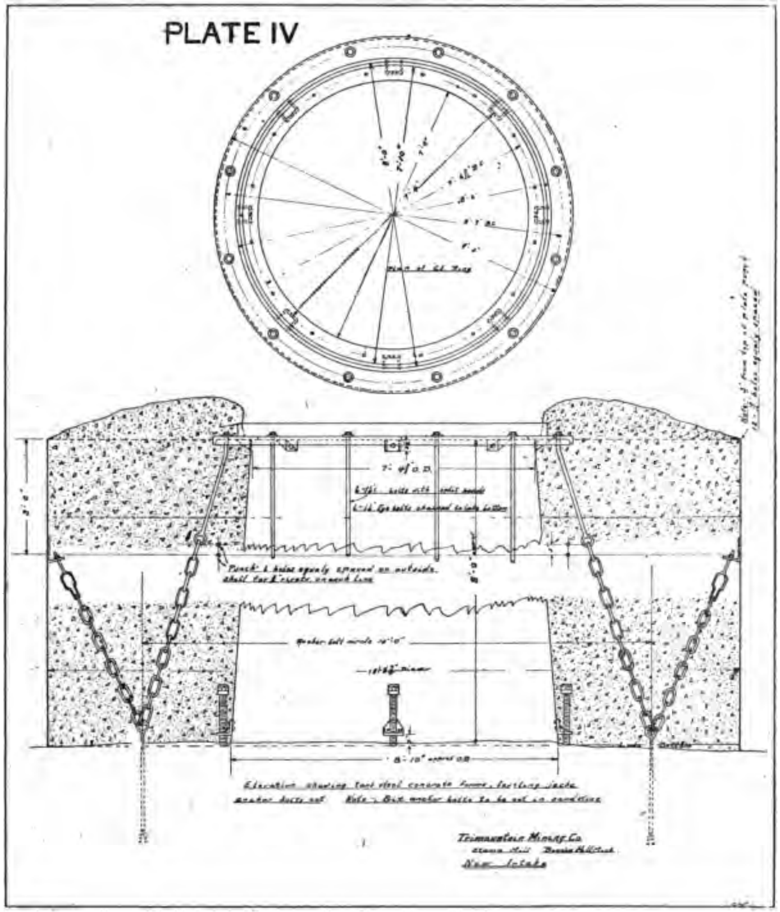


PLATE IV

put in place of the ones removed and the whole covered with concrete. This work cost $2,388.45.

On November, 28, 1911, a violent storm broke the crib away and drifted it a mile west, landing it at the Freda Park. The bottom of the crib and some ballast was left, and fortunately the pipe was not disturbed. The loss of the crib

caused no worry, as it had been a constant disturbance. Before the crib was lost it had been decided to abandon the pipe and drive a tunnel with the equipment then in use extending the Champion mill tunnel.

Trimountain's total expenditures on its pipe and crib were as follows:

Original, including 1903..........	$64,043.10
Filling crib with rock, 1905........	821.43
Placing concrete with caisson, 1907.	5,988.97
Filling crib with concrete, 1908....	1,411.01
Replacing damaged pipe, 1911.....	2,388.45
Total	$74,652.96

Ground was broken for Trimountain's tunnel intake about December 1, 1911. A shaft was sunk 100 feet below the collar or 74 feet below lake level. A pump station was cut and pumps placed at the bottom. Drifting commenced February 13. Work was in 8 hour shifts. Two drilling machines of butterfly valve type worked simultaneously, two men to a machine; also two trammers to a shift. After blasting the miners and trammers of the next shift cleared away enough to rig up. The bar was set horizontally. The miners drilled upper holes while the trammers mucked the rock. Each shift squared and left the rock for the next shift to muck.

The first 1,000 feet was dry. Then the tunnel began making water, the flow increasing to 600 gallons per minute at 1,500 feet where the face became dry. Water was struck at 1,600 feet and reached a maximum flow of 800 gallons per minute. At 1,770 the face became dry again and remained so until the end of the tunnel was reached, 1,970 feet from the shaft. Aside from the wet spots the work presented no extraordinary difficulties. The work progressed as follows:—December 1 to February 12—Equipment installed and shaft sunk 100 feet.

February 13 to 29, drifted.........	203 feet
March, drifted	394 feet
April, drifted	378 feet

(a) Caisson on Rolls Ready to Launch.

(b) Caisson in Place Over Pipe, Caisson Workers are Placing Concrete.

(c) Anchor Ice Removed From Champion Intake Screen.

(d) Siphon Cleaning Champion Intake Shaft, January 6th, 1911.

(e) Derrick Car at Work Clearing Intake Shaft at the Champion Copper Company Mill, January 17th, 1911.

PLATE V

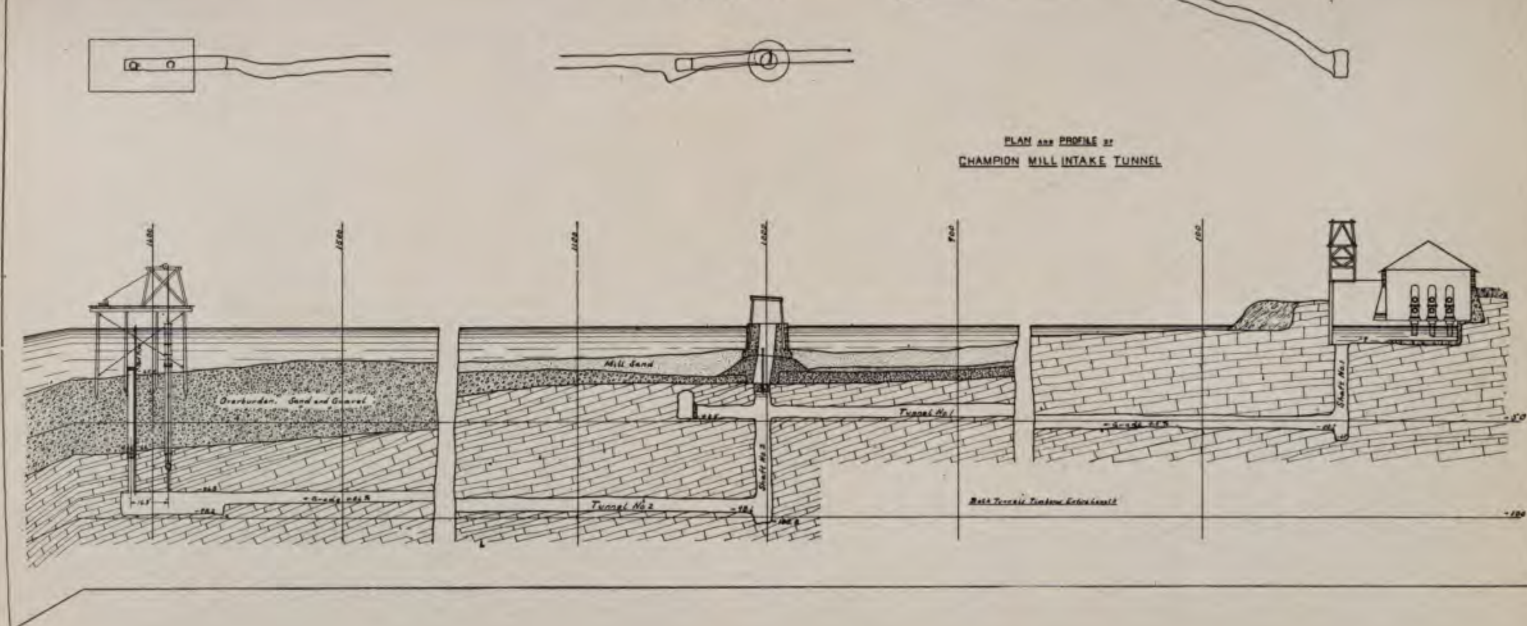
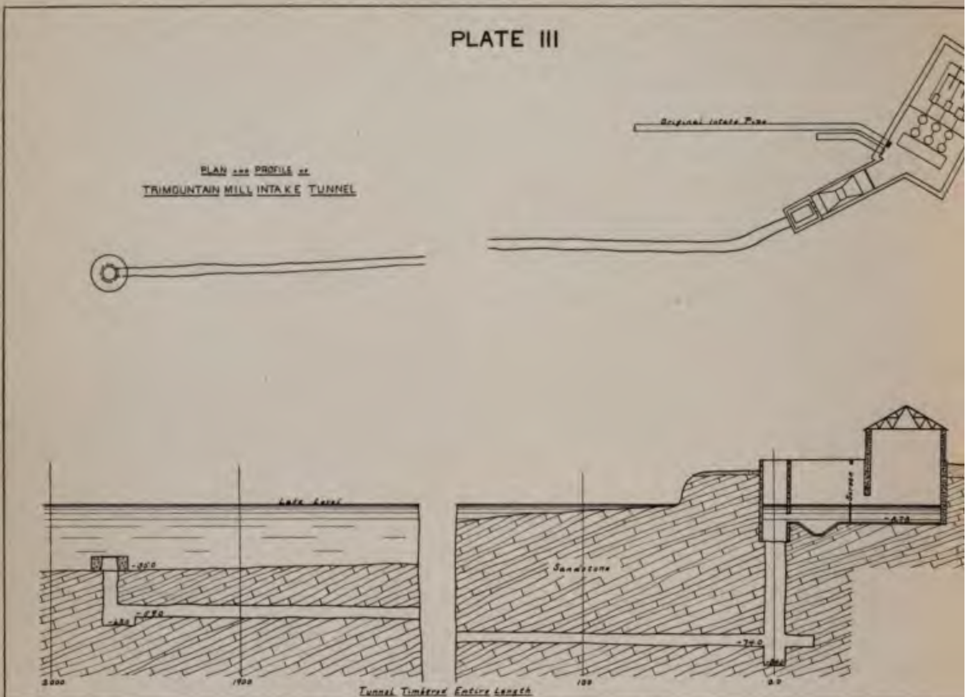

PLAN AND PROFILE OF
CHAMPION MILL INTAKE TUNNEL
Mill Sand
Overburden. Sand and Gravel
Tunnel No. 1
Tunnel No. 2
Shaft No. 1
Shaft No. 2

PLATE III



Concrete Set at the Ahmeek Mine, Referred to by E. R. Jones in H. T. Mercer's Paper "Some Applications of Concrete Underground", pages 167 to 185.
(Photo by Andre Formis)

May, drifted	262 feet
June, drifted	312 feet
July, drifted	280 feet
August 1 to 15, drifted	141 feet
Total	1,970 feet

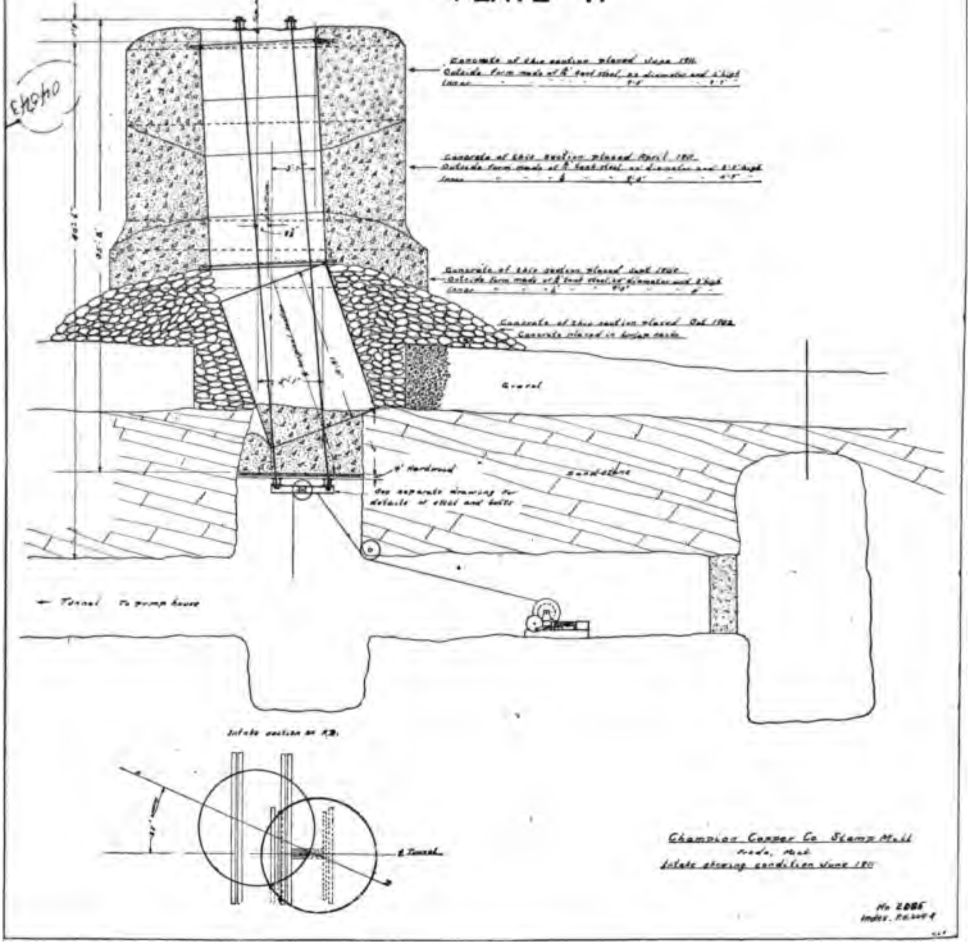
The latter part of June we drilled a hole 21 feet deep into the rock of the lake bottom, 1,970 feet from the shaft, and into it cemented a one-inch iron bar. With this bar as a center the tunnel inlet of concrete, in steel forms 8 feet high, 19 feet diameter, with a seven foot opening, was placed as shown on Plate IV. A timber covering, strong enough to keep out the water if necessary when the raise from below holed through, was placed in the position to be later occupied by the grating.

On August 15 the one-inch bar in the center of the inlet was disclosed in the breast of the tunnel. Preparations were now made to raise through to the lake bottom. A sump 6 feet deep and 20 feet long was taken out at the end of the tunnel and floored over. With this bar for a center a 7 foot raise was started and as the raise progressed the quantity of water increased. With eleven feet to go the raise was making 100 gallons per minute and the pumps were handling 900 gallons. If the flow were increased much more it would be difficult to get the pumps out. The raise was therefore stopped, the rails taken up, the floor over the sump removed and a concrete bulkhead placed in the tunnel about 20 feet from the shore-shaft. Openings were left in the bulkhead for the water to flow and a door for the men to come and go.

A hole was now drilled up into the inlet. It ran a full stream. Another hole was drilled and the two holes let in about as much water as the one had. The timber cover on the inlet apparently held out the water. A round of holes was now drilled and charged. To charge some of the holes a wooden plug was necessary with a hole in the center for the battery wires, the powder pushed up into the hole and the plug driven in with the wires projecting through it. The holes

were wired up to the electric light wires, a retreat made to the bulkhead at the shaft, the door closed and the round fired. The pumps were taking care of the water all right and an in-

PLATE VI



spection of the raise showed that the round had taken out about three feet, squaring up at a seam between two layers of rock. The two through holes were again plugged and drilling resumed. For 14 inches the drill was in very hard rock and

the hole was dry but at that depth the drill struck water and the hole ran a full stream. By moving the drill in about four inches and drilling another hole it was found that the first hole let down nearly all the water and the second hole carried up five and one-half feet, through rock that was very much softer than the fourteen-inch layer at the top of the raise. It had been planned to raise with light blasts but the large amount of water encountered above the hard layer made working in the raise so disagreeable that it was decided to drill twenty holes up five and one-half feet, also twenty holes through the hard layer for drains, load and get ready to blast, remove the two plugs from the through holes to let the inlet drain and then blast. This plan was carried out but one of the two plugs broke off and could not be removed. Some time was spent in an attempt to bore it out but was not successful with the tools at hand and further delay might jeopardize the success of the blast by chilling the powder, the water having a temperature of 43 degrees F. Staging and tools were removed, the air pipe secured and the door in the bulkhead closed. The blast was set off September 5.

The force of the blast instead of throwing the material down into the sump was turned upward by the hard layer of rock, knocking the timber cover off the inlet and cracking the concrete, also shearing off some of the bolts in the steel forms. The diver found the hole in the inlet half filled with broken rock through the crevices of which the tunnel had filled with water. A little of this rock was removed by him while the shaft was being cleared of pumps and timbering. On September 14 the concrete bulkhead was blasted out. The water raised in the shaft eight feet above lake level after the blast. Force enough had been transmitted through the water in the tunnel for about 2,000 feet to dislodge the broken rock at the inlet of the tunnel, the inlet being found clear of rock by the diver at his next descent. While the damage to the concrete inlet is to be regretted it was not ruined. After bolting the screen in place the lake end was considered finished

The two-inch air line was left in the tunnel to remove any anchor or frazile ice that might form on the intake screen during the winter. The wall into the pump well was cut and arched over on Sundays and the tunnel put into commission October 7, about ten months after the ground was broken for the shaft, at a cost for labor and supplies as follows:

	Labor.	Supplies.	Total.
Surface equipment	$ 713.24	$ 325.43	$ 1,083.67
Shaft sinking	3,816.07	1,807.78	5,623.85
Shaft timbering	456.82	184.96	641.78
Shaft concrete	489.85	235.10	724.95
Tunnel drifting	17,312.07	5,481.75	22,793.82
Tunnel timbering	1,593.09	383.10	1,976.19
Pumping	1,384.17	2,282.17	3,666.34
Intake	1,013.79	331.02	1,344.81
Channel to pump house	1,586.40	1,069.04	2,655.44
Grand totals	$28,365.50	$12,100.35	$40,465.85

Note—Machinery and rails were taken from the mine equipment and returned. Maintenance and freight only appearing in the supplies.

Plate I shows a plan and elevation of the forty-inch intake pipe as it was installed in 1901-2.

Plate II shows condition of pipe in 1909.

Plate III shows a plan and profile of intake tunnel completed October 7, 1912.

Plate IV shows the construction and method of anchoring the concrete inlet at the lake end of the tunnel.

The water supply at the Champion Mill is through a tunnel intake, the construction of which has been very ably described in a paper presented to this Institute in August, 1903, Vol. IX, p. 127, by Mr. F. W. O'Neil. No trouble of note occurred until the winter of 1905, when the supply was shut off almost entirely for thirty hours. We thought fine sand and chips had collected in the bottom of the shore shaft and closed up the tunnel portal. Soundings showed a deposit there, but we had inadequate means to clean it out. An iron

two and one-half inches square by ten feet long with prongs, was attached to a rope and plunged repeatedly into the accumulation, the pump keeping the water in the well as low as the suction pipes would permit. Finally the tunnel contributed its usual flow of water. Later an orange peel bucket removed the accumulation, but the amount did not seem enough to have stopped the flow. We suspected that during the agitation some of it had been sucked by the pump into the mill circulation. Several stops occurred that winter of two hours and longer. During the summer there was no trouble but each succeeding fall and winter brought a little more sand and wood chips to the shaft, increasing as the waste sands encroached on the outer inlet. It was finally observed that the water in the well was lower while the pump was in operation than formerly.

In 1908 we operated a clam-shell bucket every Monday morning, the cost being charged to operating and not to the tunnel. In 1909 a diver inspected the outer inlet. The tailings from the mill had filled in as high as the concrete collar around it, which had been built up four feet above the lake bottom. The screen was removed from the recess in the cast iron ring over the inlet, a steel shell of ¼-inch plate, 9 feet in diameter and four feet high was placed in this recess and a second ring placed on top of it, both shell and ring securely held by extending the original anchor bolts. Another ring of 3-16-inch steel plate three feet high, 25 feet in diameter, was placed on the bottom as nearly concentric with the first ring as possible. The sand between the two rings was removed with an ejector operated by a water jet with a pressure of 160 pounds, using fire hose to conduct the water to the ejector. The space was then filled with concrete which was mixed on shore, and conveyed to the intake on a scow, then lowered with a self-dumping bucket that held about 8 cubic feet. Care was taken to lower the bucket to the bottom before dumping. This work was carried on from a raft anchored above the intake. The completion of this work cost $1,039.32.

In January of 1910 the mill was shut down for 36 hours. The lake at the time was very quiet and free from ice over the inlet, the ice field having drifted out. A boat was carried out over the ice hummocks formed on shore, (which sometimes built up from 12 to 20 feet) and with difficulty launched. The inlet screen was found covered with anchor or frazile ice from 16 to 20 inches thick; when barred loose it floated to the surface, the water draining from it like a sponge. Some of the pieces had a beautiful fern like structure, a picture of which is shown marked (c). The pump could not be kept going by barring the ice from the screen, it reforming very rapidly. A one-half inch rope dipped into the water and removed after two minutes, attained a diameter of three inches by ice needles attaching themselves to the rope and building out in radial lines. Ice forming on the screen in this manner was undoubtedly the cause of many stoppages. In August, 1910, an inspection of the inlet showed two feet of the concrete collar above the sand bottom. This was considered safe and further raising unnecessary this year.

On January 2nd, 1911, the water flow of the intake was stopped entirely, and an inspection on the lake was impossible owing to the prevailing storms, but it was surmised the inlet shaft was partially filled with sand. The pump not operating the two preceding days, it was considered that very little sand was drawn into the tunnel, therefore, lowering the water in the shore shaft might cause the sand to flow toward and be taken out at the shore sump. A No. 11 Cameron pump was suspended in the shaft and the water lowered 38 feet but no water came in the tunnel. The weather in the meantime had moderated, making it possible to inspect the lake inlet from the ice field by placing planks to walk on. Soundings indicated two feet of sand over the inlet. A pointed pipe could be forced down its full length, from which it was concluded that a lot of the filling was slush ice heavy with sand. This proved to be true. A platform was placed on the ice, and a two-inch steam line run from the boiler house to the inlet. A

four-inch pipe had already been forced down the intake shaft about 18 feet, into which the two-inch steam line was turned. Steam was admitted and this thawed some of the slush ice, making a depression in the sand over the inlet. It was then decided to use the four-inch pipe for the suction, of a four-inch steam syphon, and discharge the sand from the shaft. This proved so effective that another syphon was rigged up and operated. The syphons in operation are shown in picture marked (d). Arrangements were being made to run the syphons continuously, when someone reported a crack in the ice field near shore. Bars were driven into the ice and an attempt made to hold the field by extending ropes from them to the shore. We managed to save the syphons and the steam pipe, but the rest of the material went out with the ice. In the morning the ice field was again in shore and we made another start, but at noon the ice moved out. This time we made no effort to hold it, but saved our entire rigging. The ice stayed out a whole day and on its return we again rigged up, and had scarcely started work when the field once more drifted out. So far we had 12 feet of sand removed from the shaft, 22 feet remaining. The ice was too uncertain to depend upon and this plan was abandoned.

A pile driving derrick car was secured from the Copper Range Railroad, a track laid to the lake, a cut made through the ice hummocks, piling and bridge stringers secured. On January 13th, 1911, we drove six piles of the trestle and completed it from shore to inlet on the 16th. The ice field now seemed more regular in its habits and stayed in shore. On January 17th, the pile driver leads were removed and the derrick boom rigged with orange peel bucket. By this method the work of removing the sand from the shaft progressed very well. Picture of the car in operation is marked (e). A small stream of surface water was turned into the pump well, raising the water there about four feet above lake level, and at 7:30 p. m., this water in the pump well ran out, thereby demonstrating that the tunnel was open.

On the 27th of the following March, 1911, the inlet again filled with sand. A head frame of three-inch plank was erected over the inlet shaft as shown in Plate IV, and used to place a ring of 3-16-inch steel plate, 8 feet high by 22 feet diameter, around the inlet. The sand was removed with ejectors operated by a water jet at 160 pounds pressure, and the shaft cleared of sand by April 10th. On April 14th, a storm carried away this head frame, the legs breaking first, also crushing the steel ring. After the storm abated the wrecked 3-16-inch plate was removed and straightened. Another head frame of green birch poles was erected. The 22-foot ring was again put in place, also the inner ring, 9 feet in diameter, the space between cleaned out and concrete raised six feet below lake level. The inlet shaft was again cleaned out and on April 26, milling operations resumed.

The intake plans for the future called for a 600-foot extension to the present tunnel; a temporary water supply through an open cut in the waste sands to the pump well; and the locating of a favorable place in bottom of lake for end of tunnel by means of test pipes. The open cut was excavated by shoveling, with sloping sides to lake level. Sheet piling was then driven down 3 feet below grade, from shore to pump house, a distance of 625 feet. The work was so conducted that the heavy cut near the pump was taken out and the rock between the well and the shore line blasted out to grade, before the sheet piling was all driven. This made it possible to remove the sand inside the sheet piling below the lake level from the pump well towards the shore, permitting all the water made by the excavation to flow towards the pump well where it was removed with a syphon. This cut was started on April 11th, 1911, and on June 10th, the water from it was being used to operate the mill. The cost of this temporary water supply was $8,929.58. A stout timber bulkhead was placed in the pump well to isolate the shaft from the new open cut water channel.

The concrete collar around the inlet shaft was now raised

above lake level by means of steel rings and concrete filling, and an attempt made to pump out the tunnel with a No. 11 Cameron pump. This could not be done due to excessive leaks through the concrete collar at the inlet. It was decided therefore, to put a concrete plug in the shaft over the steel frame that was designed to carry the sheaves, to be used for hoisting the cage in shaft No. 2. Four two-inch rods carried this steel frame, shown on Plate VI. The water was kept down below the steel while two courses of hemlock plank were fitted in the rock, and concrete put in place. This was allowed to set one week. The engineers in the meantime had located a point for No. 2 shaft on our concrete collar by triangulation, plumbed down and established points on the under side of the hemlock plank for the new workings. This also gave an opportunity to close up their survey of the old tunnel after it would be cleaned out. The shaft was again pumped out with a flow of only 200 gallons per minute. This permitted an examination of the tunnel which was found one-half filled with sand, and the small amount of timbering mostly down. In some places it was down on the up stream side only, forming an excellent dam and thereby nearly closing off the water. The hanging was down three and one-half feet in some places and quantities of loose rock had to be barred down. The tunnel was timbered following the clearing and completed in August at a cost of $6,361.85, which includes cost of timbering and a charge of $1,079.10 for power.

Ten test holes had been driven in the overburden on the lake bottom at intervals extending 600 feet out from the old intake. The sandstone was found 33 feet below lake level 100 feet from the old shaft, and 66 feet below at the 600 foot point. The test holes between the old shaft and the 600 foot point, indicated an even grade. Soundings at the 600 foot point showed a depth of water 26 feet, and sand and gravel overburden of 40 feet. A hole here was carried down in the rock 23 feet and an iron bar cemented in. This hole showed that it would be necessary to sink shaft No. 2

to a depth of 103 feet. This was done at the end of the old tunnel, and was made large enough for a cage with a nine foot tram car and a ladder way. The work was done in eight hour shifts, with two drilling machines, two miners to each drill and four muckers, the muckers working ten hour shift. Sinking of this shaft was commenced Sept. 1, and completed Oct. 9th, at a cost of $3,937.90 including timbering and a charge of $597.30 for power, as well as all labor for installing hoisting engine, etc.

Drifting the No. 2 tunnel was started October 9th, and driven at the following rates: During October, 168 feet; November, 301 feet; December, 131 feet, reaching the iron bar on the 15th. This part of the work was completed at a cost of $8,691.14 including timbering and power cost of $1,079.10. A sump at the end of the tunnel was taken out 5 feet deep and fifty feet long. Fifty feet of the new tunnel was concreted owing to the rock being very badly shattered. The timbering was completed, rails taken up and hoisting engine and pumps removed by January 14, 1912, at a cost of $3,081.41. Before the tunnel was allowed to fill the air line was well secured, the end raised nearly to the timbering and fastened there. This was for the purpose of removing any anchor ice or other obstruction that might adhere to the screens on top of inlet pipes. The rails in the upper or No. 1 tunnel were now removed. In order to use the old inlet and tunnel until such a time as the new inlet pipes could be placed, it became necessary to remove the concrete plug heretofore mentioned in old shaft. Eleven holes were drilled into it and preparatory to blasting, the tunnel and shaft were cleaned of everything except a No. 11 Cameron pump. Then the eleven holes were blasted from above with a battery. From the known leak in the concrete collar it was estimated that the tunnel would fill with water in about four hours, but after blasting, it actually filled—both tunnel and shaft—in five minutes. Where the water came in so quickly will be shown later. Our mining captain and his

force were then transferred to the Trimountain Stamp Mill where a shaft for a new tunnel was well under way.

Keeping the open cut clear at its inlet until the ice formed on the shore was the most disagreeable task we were saddled with. The shore line on the waste sands from the mills would shift with every storm, having been known to recede as much as 150 feet after a very heavy storm from one direction; a storm from another direction would partly build it up again, in any case throwing up heavy sand bars. To make provision against the fall storms so prevalent here, 200 feet of 48 inch riveted pipe was built of ¼ in. steel plate with heavy re-enforcing angles every 12 feet on the outside to help it withstand the crushing action of the ice.

The inlet to the open cut had been maintained 8 feet wide and 2 feet deep, but it was now cleaned out to a depth of 4 feet. The 200 feet of pipe had been previously bolted together on skids parallel with the shore, both ends being closed with wooden blind flanges. When the weather was favorable the pipe was rolled into the lake and floated into the open cut. The spreaders that kept the sheet piling apart were removed one at a time until seventy feet of the pipe was in the cut, when the sheet piling showed signs of giving away. The pipe was then allowed to fill and sink in place, and this seventy feet of the cut filled in. During this fall the storms were unusual and the shore line receded fully 150 feet, and it was found necessary to add 30 feet of pipe in the cut. When this addition was made, there was not velocity enough in the pipe to carry the sand through which accumulated, thereby cutting off the water supply. This made it necessary to blast off seventy feet of the outer end where the trouble was caused. The pipe then scoured itself clean with the head we had available. A shelter was built over the inner end of the pipe in which a fire could be built, to make the place warm enough for the crew that was kept busy days and some nights removing the sand carried in through the pipe. A three inch water main was laid to the shore, on which was maintained

a pressure of 160 pounds, this water was used to operate two ejectors made of 4 inch tees and 3 inch pipe, each ejector had a gate valve in the discharge pipe. This valve was closed turning the water back through the suction pipe when a stick or other obstruction chocked the tee, which occured very often. To keep this cut open and install the 48 inch pipe cost approximately $4,802.92 covering a period of nine months while the canal was in use.

During the month of February, 1912, the cause of the tunnel filling so rapidly after blasting out the concrete plug was discovered. As shown on Plate VI the concrete collar around the inlet was raised four different times, the first time in 1902 when the opening was made into the tunnel, the second in September, 1909, the third in April 1911, the fourth and last time June 1911. After placing the outer shield in April 1911, for the concrete form, on the sand around the inlet, it was necessary for the diver to remove the sand down to the old concrete. This sand had been handled so much during the past winter and spring, that a concentration of slime and pieces of water logged wood had taken place around the inlet, thereby making it very difficult to clean the place out. As soon as the diver thought he had enough of the old concrete exposed to form a base for the new concrete to rest upon, it was covered by small sections at a time. By this process it was expected that the sand would only be in small pockets and so retained by the concrete. This was true when we pumped out the shaft, no excessive leaks showing. After the plug was put in the shaft, the wave action scoured out the sand spots in between the 1909 concrete and the concrete placed on it. To place the inner steel shield in position the bottom of it was made 1½" less in diameter than the ring already in place. This allowed free access of water and possibly sand. In the spring of 1912 to secure the shaft from this temporarily, the diver drove pine wedges all around, making the inner lining practically sand tight. About one third of the fourth section of the con-

crete collar was now taken off to allow the water to flow into No 2 shaft, and the open cut was discontinued.

In May, 1912, ½x74x118 inch steel plate was purchased and 2 pipes 36 inches in diameter, each 43 feet 4 inch long were made. These were fitted with steel flanges on one end and steel drive shoes on the other. Longtitudinal and butt straps were placed on the inside, and all rivets countersunk on the outside. Four sections were also made, each 6 feet 2 inches long, one of which had a steel flange on one end. Each section had the straight seam riveted and the butt strap riveted on one end, so the four sections could be slipped together and then bolted. With these provisions one section could be made from the four, 24 feet 8 inches long, and bolted to either long section. While the pipe was being made, eight piles were jetted down in the overburden on the lake bottom over the end of the 600 foot new tunnel, and a platform was erected on them, 53 feet by 27 feet, and ten feet above the lake level. A head frame 20 feet high was placed on this platform, also the 5x8 inch Lidgerwood Friction drum hoisting engine used in the tunnel to hoist the cage in the No. 2 shaft. A five inch water main was laid on lake bottom from the shore to the platform, also a 1½ inch air line to operate the engine. Air was furnished by the compressor used for tunnel purposes. The five inch test pipe left in the lake bottom was extended to the water level and a set of blocks and tackle suspended from the head frame just over the point located for the 36 inch intake pipe. After clearing the top of the overburden of all large boulders which were lying at this point, one of the 43 feet 4 inch sections of pipe, with ends blanked, was rolled into the water, towed out and fastened to the suspended tackle blocks, pulled up and the bottom head removed. When the pipe was lowered to the lake bottom it stood plumb and in proper place. In this position a frame or guide was made for it at the top through which it could move down. The top flange was removed and three 2½ inch water jets were put in. The jets lowered the pipe 15 feet in 15 minutes, the

water being turned off when the flange neared the water level. Two of the short sections were then bolted on, the pipe loaded with six tons of old stamp shoes, and the water again turned on. The pipe went down fairly straight, but if showing any tendency to list, the tackle was held, and the pipe suspended in the boil until it became plumb again. The water washed out all the fine sand, while the gravel and large boulders accumulated at the bottom. To remove the coarse sand and gravel a six inch air lift was used, while the large boulders, some of which weighed 300 pounds, were taken out by the diver.

The pipe was then driven with a 2,000 pound weight, the cleaning and driving continuing until sandstone was reached. A 12 inch pipe with guides to centre it was then lowered down the large pipe. Into this a single bit, 12 inch drill, with stem rising to the platform was introduced to drill a pilot hole to the tunnel. This was followed by a special reamer 34 inches in diameter, made of cast steel with 12 taper holes drilled into the bottom, the holes being fitted with twelve steel cutters. The hole was churned with this reamer, by using the friction drum of the hoist to actuate it, a 14 inch drop being maintained. By this method one hole was drilled and reamed in five days. After the drilling was completed the air lock from the Trimountain Mill caisson was fitted to the top of the pipe. A pneumatic chipping hammer was used to trim off the rock on the sides for about two feet to clear the steel shoes, and the 36 inch pipes were then driven into the rock. Screens 12 inches high were placed on each pipe.

This work on the intake pipes cost for labor and supplies $5,565.07.

The Champion Copper Company has been in operation on the shores of Lake Superior for ten years and during that time they have spent for water supply as follows:

	Labor.	Supplies.	Total.
Cost of the first intake tunnel....			$26,389.62
Extending Tunnel Itemized Below—			
Raising outer shaft 4 feet during September, 1909	$ 610.17	$ 429.15	$ 1,039.32

Cleaning out intake shaft during January, 1911	1,561.71	588.65	2,150.36
Raising outer shaft 4 feet 9 inches during April, 1911	2,281.39	573.37	2,854.76
Cleaning out intake shaft during March, 1911	395.45		395.45
Construction of temporary intake canal	6,025.70	2,903.88	8,929.58
Raising outer shaft to lake level, 9 feet, 3 inches, during June, 1911, and putting in concrete plug in shaft	1,566.00	785.41	2,351.41
Constructing head-frame, engine house, placing engines, compressor, piping etc., preliminary to tunnel work...........	2,655.00	833.86	3,488.86
Cleaning out and timbering old tunnel, June, July, August....	4,304.27	2,057.59	6,361.86
Sinking and timbering outer shaft, and drifting 20 feet....	2,908.80	1,029.10	3,937.90
Drifting tunnel, October 9th to December 15th, 1911.........	6,428.36	2,262.78	8,691.14
Finishing sump, tearing out track, etc..	2,980.75	100.66	3,081.41
Reopening old tunnel at outer shaft, February, 1912	456.95	14.84	471.79
Sinking two 36-inch pipes through 43 feet overburden, and drilling two 34-inch holes through 20 feet sandstone into tunnel, May, June, July and August, 1912	3,661.13	1,903.94	5,565.07
Sundries—			
Maintenance of intake canal while in use and laying large intake pipe	1,821.87	2,981.05	4,802.92
Taking soundings, locating sandstone, end of tunnel, etc......	1,007.65	637.38	1,645.03
	$38,665.20	$17,101.66	$82,156.48

Plate V shows the old Champion tunnel with the inlet sealed up and a 12-inch pipe in the seal, put in to clean out No 2 shaft when necessary. It also shows the 600 foot extension with the drilling rig in place.

Plate VI shows the work done on the old inlet in detail, also hoisting engine and head sheave in place.

The work described in this paper has all been done by the companies' employes working under the direction of their foremen. The diving and work under compressed air was done at the Trimountain Mill by Oscar Johnson, Master Mechanic, and at the Champion Mill by Herman Stealow, Surface Foreman. The mining work has been under Captain John O. Peterson.

DESCRIPTION OF AN AIR BALANCED HOISTING ENGINE—FRANKLIN MINING CO.

BY R. H. CORBETT, HOUGHTON, MICH.

This hoisting engine was built in conformity with the ideas of R. M. Edwards, General Manager of the Franklin Mining Co., and the other Dow properties in the Lake Superior copper district. Mr. Edwards has long held the opinion that for deep mining operations a single compartment shaft would be desirable, provided that a balanced hoisting engine could be built to operate a single skip or cage. He considers that the simplicity of a single compartment shaft and its equipment will warrant any complication of the hoisting engine necessary to produce the desired result. It was apparent to him that if the power developed by the decending skip could be applied to compressing air it would furnish a solution of the problem.

When the time arrived for ordering a hoist, the matter was placed in the hands of Mr. Bruno V. Nordberg to design an engine and work out the details of an air balanced hoist to meet the requirements. Following this the engine under discussion was built by the Nordberg Mfg. Co. of Milwaukee, Wis. This hoisting engine is a horizontal duplex machine with Corliss steam cylinders attached to the frames. The air compressing cylinders are located immediately back of the steam cylinders. The air pistons are attached to extensions of the steam piston rods, the whole forming a complete hoisting engine and air compressor combined. Means are provided for allowing the steam pistons to run free while lowering the skip. The air pistons run free while hoisting. The engineer regulates the speed of the decending skip by

controlling the quantity of air compressed. By means of the operating lever on the platform he absolutely governs the work done in the air cylinders between the limits of full cylinder capacity and no load.

The air compressing cylinders, Fig. 1, have four corliss valves. The two lower ones "A" admit free air to the cylinders in the usual way.

The two upper ones "B" are provided especially for regulating the capacity and are the ones controlled by the en-

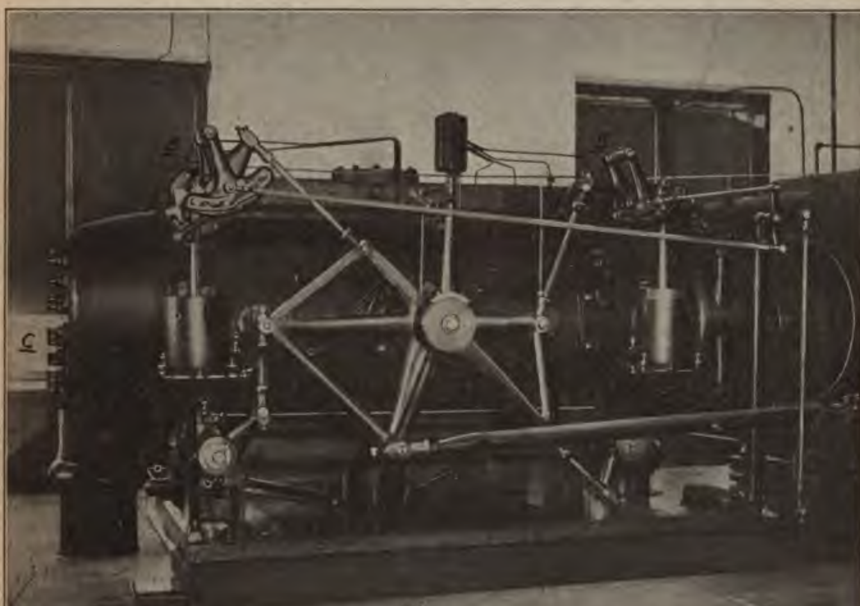


Fig. 1.
Air Compressing Cylinder

gineer while lowering. The air delivered under pressure passes through spring loaded discharge valves "C" located in the cylinder heads. The regulating valves "B" are fitted with releasing mechanism and dash pots like a corliss steam engine. When they are open they afford a direct passage between the bore of the cylinder and the air inlet pipes through

the channel "D" on top of the cylinder casting. Therefore no air can be compressed while they are open.

To illustrate their action we will suppose an air piston is moving toward the end of the cylinder while lowering the skip. If the regulating valve is open the air in front of the piston will be forced back into the inlet pipe. If however the engineer trips the valve and the dash pot closes it at any desired point in the stroke, then the air left in the cylinder will begin to compress until it finally passes out through the discharge valves. The arrangement is such, that the further the engineer moves his lever the more air will be compressed and the resistance increased on the air pistons. The air cylinders are only used for regulating the speed while lowering. The usual steam operated brakes are provided for landing the skip.

To enable the steam pistons to run free while the engine is lowering, the exhaust valves on the steam cylinders are arranged to be released from their connections and remain wide open and stationary until they are hooked up again when the engine is reversed for hoisting. The releasing and hooking up mechanism is connected to a small steam operated thrust cylinder provided to furnish power for this purpose. The operating valve on this cylinder is attached to the reversing gear of the hoisting engine. Therefore when the engineer reverses his engine the exhaust valves are either released or hooked up again as the case may be, without further attention on his part.

While compressed air is usually in demand around a mine, it was decided in the present instance, that it would be best to mix it directly with steam from the boilers and use the mixture in the steam cylinders for hoisting the load. Three large cylindrical drums were installed for storing the compressed air along with the steam. They are each 10 feet in diameter and 32 feet long. They furnish ample receiver space at the present time for both steam and air.

Fig. 2 is a diagram of the general arrangement showing how these receivers are connected with the boilers, air com-

pressors and steam cylinders. A 12 inch steam pipe connects the boilers with one end of the receivers. A 16 inch pipe from the opposite end of the receivers is carried to the throttle valve. The 12 inch discharge pipes from the air compressors are connected to this same 16 inch line. This forms a convenient way for the compressed air to reach the receivers when lowering and also to supply the steam cylinders with pressure for hoisting. A reducing valve is placed in the 12 inch steam pipe between the receivers and boilers. It is set to maintain about 75 pounds pressure on the receivers. The boiler pressure is usually about 125 pounds.

In explaining the operation of this feature of the hoisting engine, we will say that in the first place the receivers are filled with steam at 75 pounds pressure. The skip is then lowered into the mine. The air compressors begin to discharge compressed air into the receivers to mix with the steam they already contain. By the time the skip reaches the bottom of the mine the pressure will rise to 95 pounds, the increase in pressure being due to the compressed air forced in by the decending skip. They will then begin to hoist the load and have 95 pounds pressure to start with. After the skip starts upward the pressure will gradually drop to below 75 pounds as the engine uses up the air stored in the receivers by the previous trip downward. The reducing valve will open after the pressure drops and steam will be taken directly from the boilers to complete the trip upward. When the hoist stops the pressure will rise to 75 pounds through the reducing valve and be in readiness for the next trip down.

The engineer handles three levers in controlling this hoist, the throttle lever, reverse lever and brake lever, the same number as on other hoists in this section. The throttle lever however usually stands in a vertical position when the hoist is stopped. If he pushes it from him it operates the throttle valve, if he pulls it toward him it acts on the regulating valves on the air cylinders. With this exception the hoist handles about the same as other hoists in the copper country.

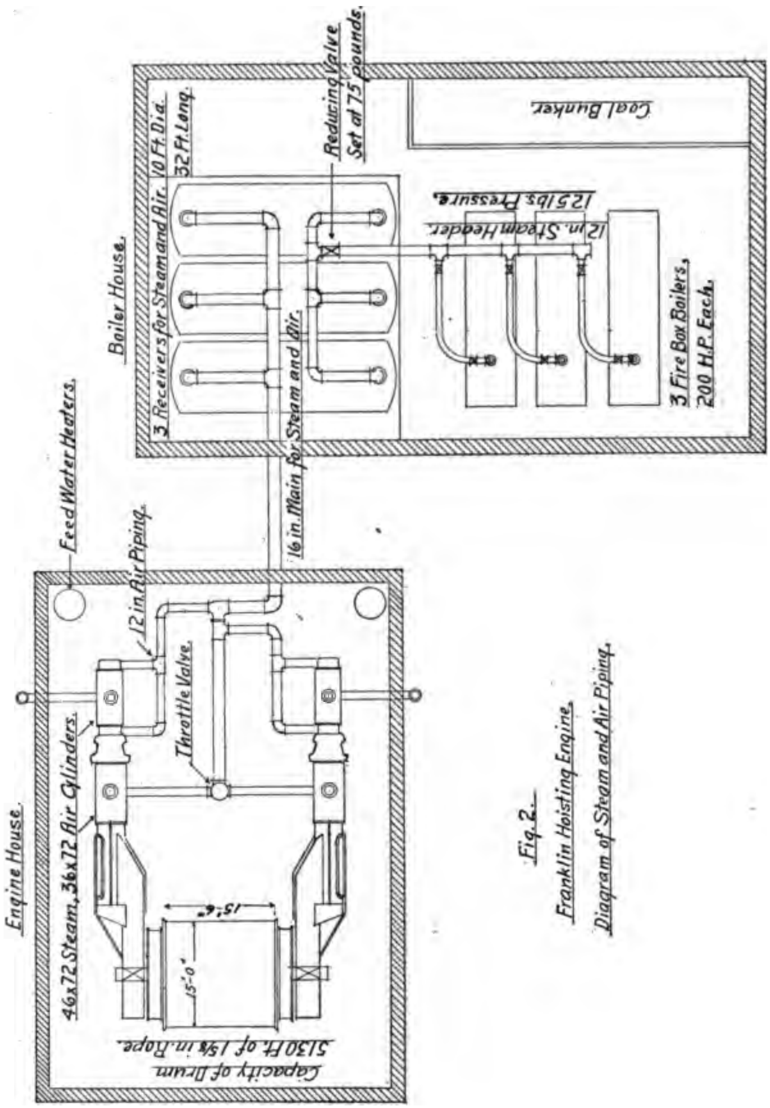


Fig. 2.

Franklin Hoisting Engine.

Diagram of Steam and Air Piping.

GENERAL DIMENSIONS AND DATA.

Diameter of steam cylinders....................46 in.
Diameter of air cylinders.......................36 in.
Stroke of all cylinders ....72 in.
Diameter of piston rods.........................7½ in.
Size of crank pins..........................12x12 in.
Size of cross head pins.....................8⅝x14 in.
Size of main bearings.......................20x40 in.
Diameter of hoisting drum..................15.0 feet
Length of hoisting drum....................15.5 feet
Capacity of drum..............5,130 feet of 1⅝ inch rope
Weight of skip........................14,000 pounds
Weight of rock........................20,000 pounds
Weight of rope per foot.................4.15 pounds
Number of boilers required............................2
Capacity of each boiler................200 horse power
Number of feed water heaters..........................2

(Berryman heaters now being installed).

The boilers are the Lake Superior fire box type with crown and arch tubes.

Shaft dips 47 degrees from horizontal.

Present depth of shaft ....3,400 feet
Approximate velocity of skip, hoisting....1,500 feet per min.
Approximate velocity of skip, lowering....2,000 feet per min.

ROCKHOUSE PRACTICE OF THE QUINCY MINING COMPANY.

BY T. C. DESOLLAR, HANCOCK, MICH.

The Lake Superior methods of handling copper rock, as it is locally termed, have undergone many improvements during the past few years, both with respect to underground and surface operations. It is the purpose of this paper to describe briefly the form and mode of operation of one of the combination shaft-rockhouses, commonly known as "Rockhouse." As the various rock-handling systems in the district work toward the same end—that of crushing copper rock—it is only the intermediate steps that differ from one another in the rockhouses of the various companies. It is thought that the following remarks relating to rockhouse practice at the Quincy No. 2 rockhouse might be of interest as illustrating a very economical procedure and the best method that has yet been devised by the Quincy Mining company. Past experience has shown that it is advisable to break the rock to a size that can be advantageously fed into the steam stamps at the mill. The material, as it is hoisted to the surface, comes in sizes varying from fine material to large masses of native copper, copper rock, and poor rock, weighing at the maximum several hundred pounds.

GENERAL DESCRIPTION OF BUILDING.

The building is of steel, with corrugated iron roof and sides, and re-inforced concrete foundations and floors, thus making the building absolutely fire-proof. All the bins are of steel and are circular in transverse section. The bottoms of the bins, instead of being built up and costly to maintain, are filled with poor rock until the latter assumes an angle of

natural slope, thus making its own bed and causing all copper rock dumped into the bins to discharge through the regular openings. The building is 150 feet long, and 30 feet wide except where the large stamp rock bin, 44 feet in diameter, is located. The crusher floor, situated on top of the stamp rock bin 45 feet above the discharge aprons, is of the same size, but square. This bin will hold upwards of 2,000 tons. This large capacity takes care of variations in car supply and rate of hoisting.

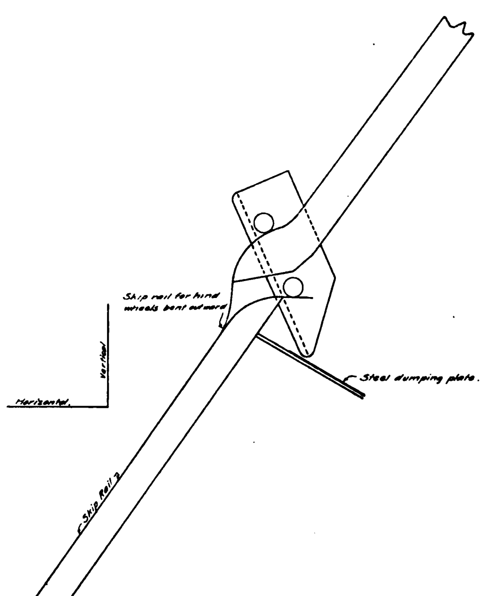


FIGURE 1.

DEVICE FOR DUMPING COPPER ROCK.
QUINCY MINING CO.-HANCOCK MICH.
1912.

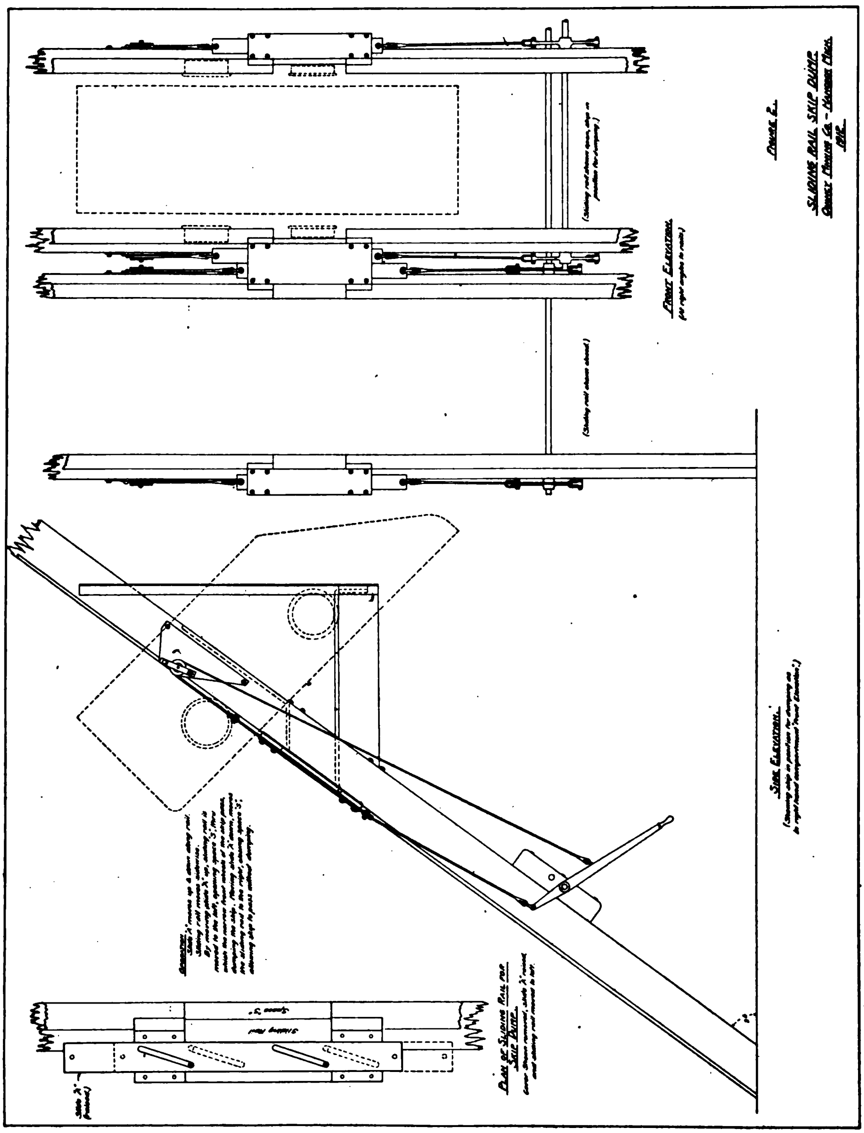
The skip track above the collar of the shaft has an incline of 54 degrees, and the shaft runners are carried up on this angle to the copper rock dump, where there is an outward curve toward the hanging of the shaft for the wider face of the rear skip wheel to travel upon, as can be seen on draw-

ing, Fig. 1. The 12-foot wood-filled rope sheaves are located 119 feet vertically above the collar of the shaft, and are securely stayed by means of a batter brace. A room located at one side of the collar of the shaft, is occupied by the lander, one of whose duties it is to transmit to the hoist engineer the signals as communicated to him from the underground chute men. The lander without leaving his room moves a sliding plate by means of levers which allow an opening for the narrow face of the front skip wheels to pass through onto a track toward the foot of the shaft, the rear skip wheels, which have a wider face, continuing in the plane of the incline, and thus dumping the skip. This dumping device is shown on drawing Fig. 2.

There are three points of discharging: The first for mass copper and dull drills; the second for poor rock from sinking shaft, or bottom level cross-cuts; the third, the regular copper rock discharge. A set of levers, one for the first and one for the second, will permit the opening of the discharge as indicated by the signals. An 8x8 inch steam hoist, located in one corner of the building, is for handling timber, cranes and other supplies; also for the track forms used to take a damaged skip or other conveyor, from the skip road. The man-cars, water skip and rock skips are suspended from cranes and can be quickly swung into place and put in operation.

ROCK HANDLING.

The rock is hoisted in two 8-ton skips running in balance. The skips, as before stated, have three discharges. The first discharge handles the large mass copper and dull rock drills. In this dump the skip discharges onto a concrete block faced with old rails (flange up) having a slope of 30 degrees. This slope changes to level and forms a platform of the right height for loading the mass copper onto a railroad flat car. Masses weighing up to eight tons are loaded upon the railroad cars by means of 8-ton chain blocks hung from trolleys in either compartment carried by 18-inch, 55-pound I-beams extending out over the railroad track. The dull drills


SLIDING RAIL SKIP DUMP
FRONT ELEVATION
SIDE ELEVATION

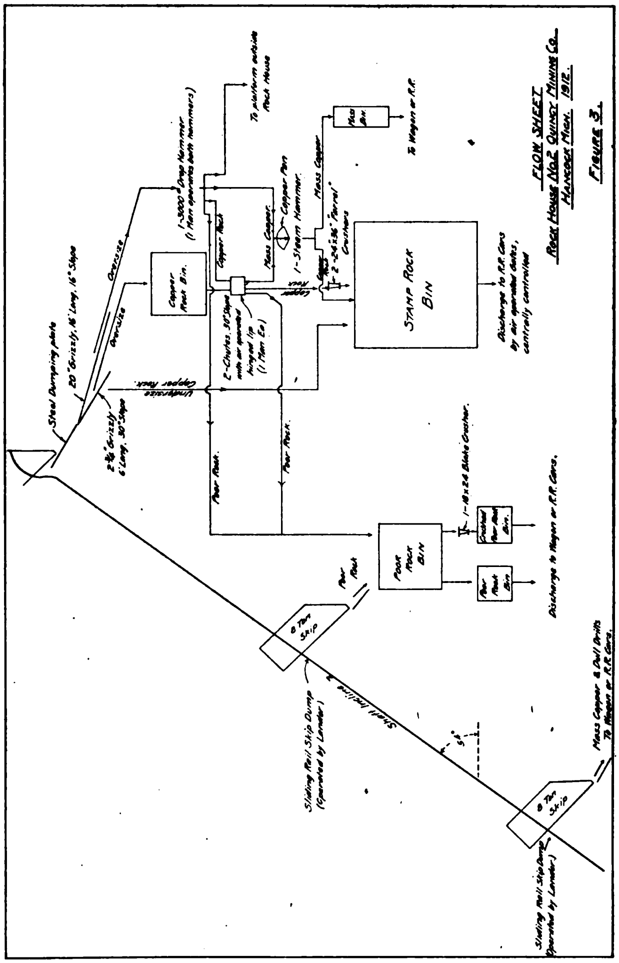
are easily loaded onto a wagon or car and taken to the drill shop to be sharpened, while the mass copper is shipped directly to smelter.

When poor rock is hoisted from shaft sinking or the bottom level cross-cuts, the second dump is opened by means of the above described levers and the skip discharges into a reinforced concrete chute, which empties into a cylindrical steel bin 13 feet in diameter. This rock is drawn off by two chutes, one of which feeds a 24x18 inch Blake type rock crusher, where the rock is crushed to a size adapted for concrete, or road work. Under the crusher is a cylindrical steel bin 9 feet in diameter, where it can be loaded from a chute into railroad cars or wagons. The chute on the opposite side of the 13-foot bin permits the discharge of the coarse rock directly into a vertical cylindrical steel tube 5 feet in diameter, dropping upon a reinforced concrete block slightly inclined, which breaks the fall, and from which it can be loaded through a chute into railroad cars, or wagons.

When copper rock is hoisted, the third, or regular, discharge is used and was made as above described in order that the lip of the skip might remain in a position close to the steel dumping plate and not cause the rock to be violently thrown from the skip. It might be stated that the copper rock dump was designed in order that the skip should discharge into a pocket which would fill until the rock took its natural slope, the rock then discharging onto the grizzlies. It was found during the erection of the building that there was a mistake of about three feet between the blue prints and the steel construction, and a steel dumping plate was substituted.

The dumping plate spreads the rock upon 6-inch round steel grizzly bars approximately 16 feet long, set at 16 degrees, and having 20-inch openings. Immediately above the grizzly bars is a battery of heavy bars, which serves the double purpose of breaking the fall of the rock and spreading the same upon the grizzlies. The oversize from the grizzlies passes down on a reinforced concrete chute, striking a second battery of bars.

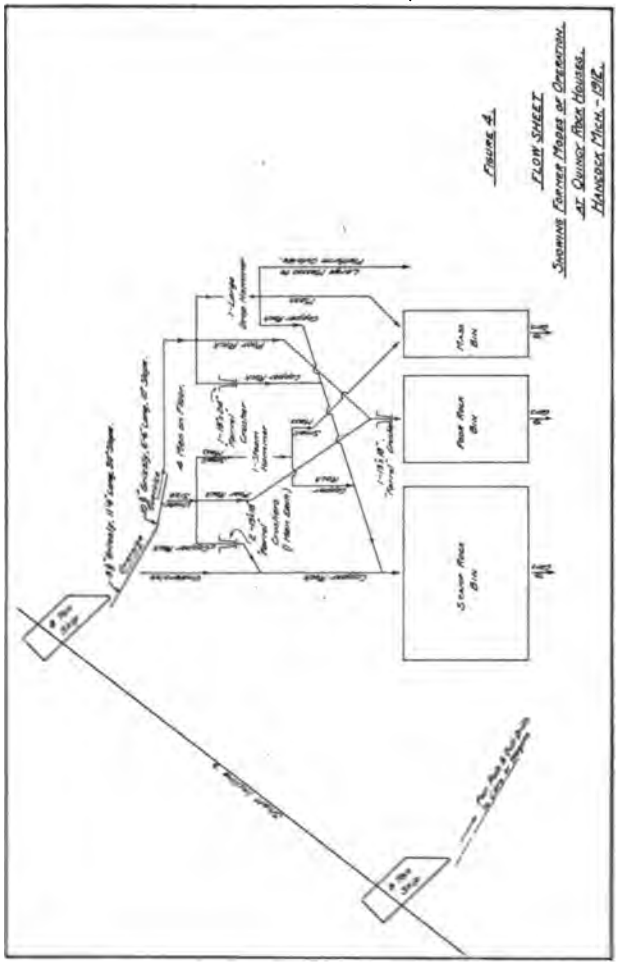
then drops vertically into a bin having its side open toward a 3,000 pound drop hammer. Between bin and hammer is a 15-inch, 42-pound I-beam carrying a traveling 8-inch 2-ton air lift. This I-beam is bent in the shape of a horse-shoe and per-



mits the air lift to be used at either crusher, or poor rock chute. Here the oversize, if mass copper, is cleaned of the poor rock under the hammer; if copper rock, it is broken to a size that can be handled by the crushers; or, if poor

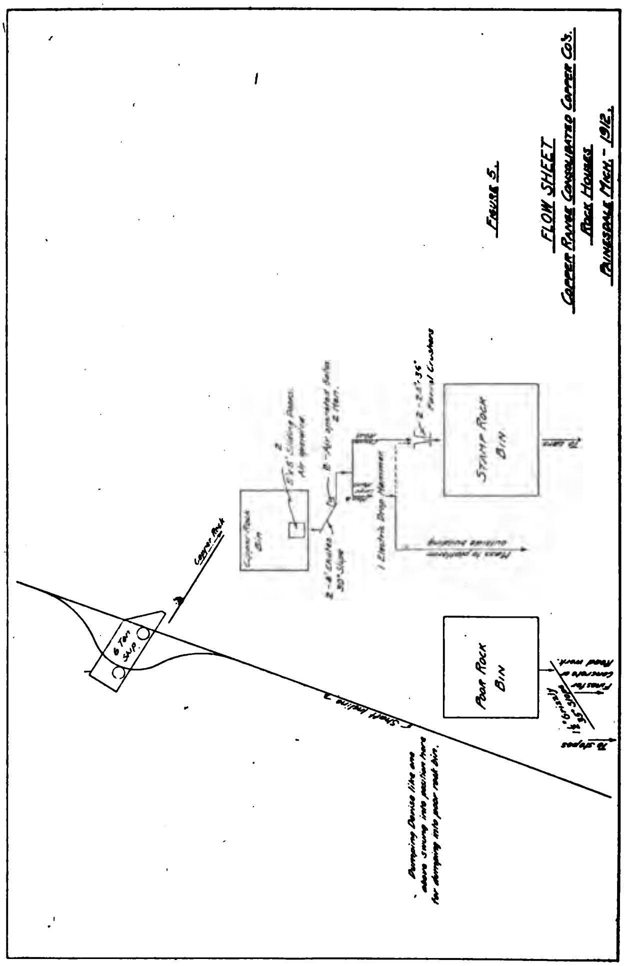
rock, it is thrown into a chute leading to the poor rock bin.

In case the broken oversize is small mass copper, it is thrown into a mass copper chute leading into a cylindrical steel bin 6 feet 8 inches in diameter, which discharges into railroad



cars. If it is mass copper too large for this chute, it is loaded onto a pan, swung from a crane, and lowered outside the building to a reinforced concrete platform at the right elevation for loading on a railroad flat car.

The undersize from the 20-inch grizzlies falls upon a second grizzly composed of $3\frac{1}{2}$ inch round steel bars, 6 feet long, set at 30 degrees, and having $2\frac{3}{4}$ inch openings. The undersize from these last grizzlies passes directly into the stamp



rock bin below the crusher floor. The oversize is discharged into a cylindrical steel bin 14 feet in diameter holding approximately 10 skip loads of rock. The outlet from this bin is by two chutes 19 feet apart. Vertically sliding steel doors, $4x4\frac{1}{2}$

feet, with replaceable steel linings, operated by 6x36 inch air cylinders, control the feed into steel chutes set at 30 degrees. This feed is directed into the two 36x24 inch Blake type rock crushers and regulated by a hinged apron, which is operated by a 6x18 inch air cylinder. One man at each chute feeds the crusher, picks out the poor rock and mass. The poor rock goes into a chute leading to the poor rock bin. The mass is stored upon an inclined semi-circular chute called the "Copper Pan," on which it easily slides to a small steam hammer, is cleaned from poor rock, and thrown into a chute leading to the mass copper bin. This steam hammer is run by a third man, who handles the trolley already described, cleans the mass copper, oils and has charge of the machinery.

The crusher jaws are made of manganese steel, and are set to crush to three inches. The crushers running at 140-R. P. M. are operated by a 12x24 inch Nordberg Corliss valve steam engine running at 110-R. P. M. A 75-H. P. electric motor occupying less space is installed, and is ready to run in case of a break-down to the engine.

The rock from the stamp rock bin is loaded into railroad cars for the mill by means of discharge aprons operated by 5x13 inch air cylinders.

Three men handle upwards of 1,000 tons of rock every 12-hour shift, change man-cars and skips on skip-road, and load all timber and supplies that are lowered underground. Between shifts, when the large mine air compressors are not in operation, power for the operation of the air lifts and air controls is furnished by a small 11x11x12 inch Westinghouse air pump, so arranged that when the air pressure from the mine compressor drops, the air pump automatically starts. A combination of check valves prevents loss of this air into the underground system and, likewise, when the air pressure from the compressor is raised, the air pump is automatically stopped. This system of handling material has shown an average rock-house cost of less than two cents per ton.

Fig. 3 is a flow sheet graphically illustrating the above description.

Fig 4 is a flow sheet illustrating the rockhouse practice of a few years ago.

Fig. 5 is a flow sheet illustrating the methods used at the mines of the Copper Range Consolidated Mining Company, as is given through the courtesy of Mr. H. T. Mercer, of that company.

IN THE LAKE SUPERIOR AREA WHAT INFLUENCE, IF ANY, DID THE THICKNESS AND CONTOUR OF FOOT WALL BEDS HAVE UPON THE SUBSEQUENT DEPOSITION AND DISTRIBUTION OF COPPER IN OVERLYING BEDS?

BY L. L. HUBBARD, HOUGHTON, MICH.

In 1898 in Vol. VI of the reports of the Michigan Geological Survey, I called attention to the evidence of extensive movements that had taken place in the Keweenaw series along the tops of conglomerate beds. The uniformly even surfaces of these beds over large areas constituted planes of weakness for the absorption and relief of stresses induced in the series by the various changes through which it was passing. To the intrusion of acid rock matter, and to the disturbed equilibrium caused thereby and perhaps also by new accumulations of lava, and of sediments, if to no more remote causes, these stresses may perhaps be attributed.

In 1904 in an article published in "Americana" I recorded the opinion that movements of a similar nature, which were on a more limited scale, had taken place in some of the amygdaloids or cappings of the melaphyr beds that are now wrought for their copper. That opinion was founded on the observation that in these beds the copper is rarely found in the rounded form of amygdules, but rather in a network of seams which extend most conspicuously in the direction of the plane of the bed and less frequently perhaps at an angle to it. I proposed for this type of Lake Superior copper deposits the term "conformable veins."

It is quite apparent that a movement in the rock complex

induced by stresses would tend first and foremost to distribute itself over the even surfaces of the extensive water-laid conglomerates and to a less extent, perhaps, through their loosely coherent masses. I here use the term conglomerate to cover also the sandstone, for they are both products of contemporaneous activities under different local conditions. Similar movements elsewhere in the series, if any, have naturally taken place in the porous cappings rather than in the more homogeneous or compact parts of the lava beds, just as pieces of steel under stress would probably first shear through a plane of blow holes. Most of the more extensive conglomerate beds if examined carefully, will probably disclose the evidences of movement over their surface by the presence of slickensides and fluccan. Not all, nor even many of the amygdaloidal beds would probably show any appreciable marks of "conformable" shearing. Why not? Anyone that has watched the slag flowing from a furnace and spreading in layers over the gently sloping surface of earlier congealed slag, will note that at times the outflowing layer freezes over its entire surface even while the furnace continues its discharges; but the flow soon breaks out at a new point and forms a new layer. Successive layers thus overlap one another, and form an imbricated complex without any very extensive planes of division. If, however, the out-flowing mass of molten material from the furnace were abundant enough, the entire surface of the slag stream might be hot enough to prevent rapid congelation, and we should see a single liquid layer covering the entire group of earlier layers, and thick enough to fill all pre-existing surface inequalities.

Some of the upturned edges of the Keweenaw flows are traceable, perhaps interruptedly, for long distances, and the beds must have been laid down under conditions very similar to those suggested above. The molten material filled first the low places and gradually spread out laterally as it increased in depth. The gases in it, relieved from pressure, rose to the top of the bed and formed its amygdaloidal cap-

ping. Here then, we had an extensive area which when it subsequently became a part of the tilted Keweenawan complex, formed a more or less continuous horizon of weakness through which shearing might easily take place, more easily than in beds of smaller area.

It has been remarked that many of our so-called amygdaloid copper deposits are immediately underlain by thick trap beds, and rarely if at all by thin ones. May not the above explain this phenomenon? As stated above, observation has already shown us that the copper in the amygdaloids is generally found not in the amygdaloidal cavities, but in brecciated material—in fissures. The thick foot wall bed rendered possible a widespread and thickly bubble-studded capping, in other words a marked and extensive plane of weakness which would be particularly susceptible to shearing. The more circumscribed the amygdaloidal capping, by comparison with the more compact rocks in the extension of its plane, the less it would be subject to shearing, and consequently the less extensively it would offer cavities of mechanical origin for the deposition of secondary minerals. At the Champion mine I remember to have seen in six years, only one piece of rock that contained shot copper.* In the Winona mine a similar condition exists. Moreover, in the latter mine a well defined slip marks the course of the lode on the hanging side, and other evidences of shearing are seen in the adjacent rock material. At the Ojibway, near the shaft that has uncovered the more copper, a fissure vein in the hanging of the east lode is in evidence from near the surface to the bottom of the mine. I say fissure vein for I believe that specimens of crystallized copper occur more often in what we are accustomed to call fissure veins than in accidental cavities of more irregular occurrence.

The effects of shearing on the volcanic rocks are various.

"*It (the Baltic lode) is no one well defined amygdaloid top to a flow but rather an impregnated shear zone or stock work, copper being found over a belt more than 40 feet wide."—Lane in Proceedings of the L. S. M. I. Vol. XIII, 1908, p. 100.

Where it has been intense the melaphyr beds are crushed and altered to a dark green chlorite. On shear planes in the latter material copper sometimes occurs in leaflets, the elusive "two-dimensional" stuff of our stamp mills. Another and more general product, in the amygdaloidal portions of the beds, is epidote with abundant quartz. All of these minerals furnish evidence of the chemical changes that preceded or accompanied the deposition of the copper. At Winona, the foot wall trap is from 40 to 50 feet thick; at the Baltic and Champion from 100 to 150 feet thick for a length of four miles; and at Ojibway and Mohawk it will probably average more than 100 feet thick, and maintains a much greater thickness for several miles. The fissure vein at No. 1 shaft, Ojibway, is perhaps not more than 40 feet from the lower side of the foot wall bed at a place where this bed was evidently much eroded in ancient times. In this case it is possible that the fissuring is transmitted, going south, to a higher horizon, a possibility we are now trying to investigate.

If the foregoing comparison and conclusion be correct, we may expect to find shearing either

(1) Limited approximately by the area of any one of these beds.

(2) Extending into or through an adjoining trap bed.

(3) Deflected into a transverse fault, or from the latter

(4) Transmitted to another amygdaloid.

Of the first of these cases we can probably collect little material evidence in this area, because we do not often mine in the traps beyond the limits of the copper. As to the second, I have sometimes thought that the Mabbs vein* may possibly be the extension of the Baltic-Superior shear zone system through the more compact trap; that the temporary disappearance of the Baltic zone of enrichment north of the Baltic mine is due to the cross fissuring and disturbance in section 16;

*On Marvine's Plate XIV a (M. G. S., Vol. I) there are shown two other faults, parallel with the strike of the Mabbs vein, at an angle of about 12° to the strike of the beds.

and that the Arcadian-Isle Royale and Grand Portage lodes come under the fourth class of a shear transmitted from one plane of weakness to another.

I think we are reasonably sure of our ground thus far, that some of the amygdaloidal beds are not only the seats of shear zones, but that the copper and some other secondary minerals in them were deposited in cavities caused primarily by shearing. Another fact of which we are sure is, that near the bottom of the Quincy mine some years ago, at a vertical depth of about four thousand feet (55th level?) a stream of mine water was found to have deposited copper in one of the newly-opened drifts. The deposit appeared to be largely an aggregate of thin leaflets whose free ends vibrated in the gentle current as it passed over them. They had a copper-like lustre, but when exposed to the air, oxidized rapidly. The writer gathered a handful of this substance which was tested qualitatively, and gave a strong copper re-action. Water afterwards procured from probably the same place, was analyzed by Ferneckes, and the result published in Vol. 13 of the Proceedings of the Lake Superior Mining Institute, page 110. It seems to have been generally believed that our native copper was deposited from solutions, and the occurrence in the Quincy mine seemed to add weight to this belief. Whatever may have been the chemical changes that led to the saturation of the solutions, there seem to be no very definite opinions as to what was the final set that precipitated the copper —and other minerals—in certain beds of the Keweenaw series, and not in others.

The various effusive rocks that compose the greater part of the Keweenawan complex, differ more or less in chemical composition, and in trying to account for the presence of copper in only a few of our amygdaloidal beds, one is prone to attribute it to their chemical character, that is, to a greater tendency to decomposition and disintegration in them than in other beds, and consequently to a freer exchange of mineral constituents between them and the permeating waters, and a

greater deposition of secondary mineral: but when we find in the Kearsarge group of seven or eight different lava beds of apparently similar character, and find copper in apparently only one or two of them, we are forced to seek some other than the purely chemical explanation. I may say here, that so far as I know, these different Kearsarge beds have not been analyzed, and some of the suggestions in this paper are offered more as a working hypothesis to stimulate further observations, than as a statement of a positive conviction. At the same time the mechanical theory appears to satisfy all reasonable requirements, and I submit it for what it may be worth. It may represent only one of several modes by which the deposition of the copper was accomplished. It is in one of the aforesaid beds at the Ojibway mine, the east lode which carries the greater amount of copper, that vein formation or shearing has been surmised.

Let us now seek to follow the copper on its way to its final resting place. It probably came up with the lavas, and at some time after they were solidified was gathered together and permeated the rock system, in the form of solutions. I say permeated, because I cannot conceive that any large mass of rock among the melaphyrs could either per se be unpermeable or remain unpermeable over a volcanic area subject to disturbances and to frequent fracturing. The solutions, where comparatively unexposed to changes from meteoric or surface waters, that is at considerable depths, must in the cycle of their chemical changes finally have reached a stage approaching chemical equilibrium or saturation, the deeper water strata probably containing the greater amount of dissolved mineral substance because of the greater pressure, and possibly greater heat to which they were subjected*. An approach to a general state of equilibrium, both in the rocks and in the permeating solutions, may thus be assumed as a possibility, for some point in time, and for the solutions may be characterized as a

*The pressure in the beds would be increased by the expansion brought about by the chemical changes going on.—Lane, ibid., p. 132.

state of depositional imminence. At some such time a fresh outflow of lava occurs. Material is removed from under the earth's crust and piled up on top of it: the rock equilibrium is disturbed, the support has been removed from beneath the lower parts of the titled beds, and some movements begin along well defined planes of weakness. They begin at low depths and extend upwards and are possibly recurrent, that is, cumulative. In the early stages there is nothing cataclysmical about them. Owing to the drag of the overlying rock mass, a small gash develops at some point most likely in an amygdaloidal bed, and the opening thus made is in the nature of a vacuum. Surrounding solutions tend to converge from all sides on this point, and under diminished pressure begin to unload a part of their mineral burden. This precipitation should be possible not only within the actual cavity itself, but with less intensity at all other points within the sphere of diminished pressure. The solutions having dropped a part of their load might be able to re-attack and dissolve some further mineral substances in the rocks, or by diffusion gradually re-assume chemical equilibrium over considerable distances. A further extension in the crack or fissure, or the development of other gashes in the bed, forming the so-called spliced structure suggested by Pumpelly*, would be followed by similar movements of converging solutions. The final extension upwards of the cracks or gashes would probably be on an increasing scale, and an opening or openings that began as a minute crack, might in several thousand feet or even less, develop considerably in aggregate width. Hence, other things being equal, we should find more space for the deposition of copper and other minerals in these mechanically made openings as we recede from the depth at which precipitation began, and approach the surface.

While the movements of the solutions towards any particular point would be converging, the general movement in a broad sense would be upwards, that is, from areas of great-

*Michigan Geological Survey, Vol. 1, Pt. III, p. 42.

er pressure. This view of the movements of mineral-bearing solutions may tend to reconcile the opposing theories of ascending and descending solutions. Under it the idea of lateral secretion would have to be reversed, the secretion of mineral being towards a vein instead of from it.

The spliced structure above referred to was applied by Pumpelly to the ordinary fissure vein. The causes that produced it must be the same or closely allied to those that bring about shearing in the tilted beds of volcanic origin. In our amygdaloids we see these on a comparatively small scale. Near the extremity of Keweenaw Point "accommodation" appears to have been made so far as the evidence is at hand, largely in the cracks that run across the formation, some of them copper-bearing, and in the Ontonagon district the drag of the overlying beds has resulted in at least two cases in fissure veins that strike nearly parallel with, but dip more steeply, than the beds of the series. It is largely between these two areas that our productive bedded deposits have been found. May they or some of them not be due to mechanical movements very similar to those that produced the so-called fissure deposits at each end of the district?

Another matter that I wish to call to your attention is a working hypothesis founded on some observations at Ojibway. In No. 1 shaft we find several bunches or chutes of good copper ground that extend to the north with depth. Between points at about 950 and 1,300 feet respectively below the collar, the shaft passed through a zone of barren trap rock running in the same direction. This zone separates two of the principal copper areas under which we usually find a bed of so-called melaphyr- or amygdaloid-conglomerate with a fine grained red matrix. A cross section at the 800 foot level shows the cupriferous beds in two narrow U shaped basins. Is it not possible that the barren trap represents a ridge of the earlier or foot wall bed (it is like the latter in character) and that the basins flanking it are channels of corrasion made by ancient Kearsarge streams flowing northerly, and that the detrital

matter now in the melaphyr conglomerate was carried from the heads or sides of the ravines, and subsequently covered by flows that coursed down the channels and united at the foot of the dividing ridge of trap?

May we not then expect to find at a greater depth these contemporaneous flows united over a wider area and making conditions more uniform for mining? In fact should not the fissure noted, not only in the upper parts of the mine, but near the bottom as well, have its origin in the deeper seated part of the bed which should be vastly more extensive than the small area confined to the narrow channels described, and therefore more likely to be the focus of a general shearing movement? I am not familiar with the conditions in any of the other mines in the Kearsarge group, and commend to the engineers at those mines a close scrutiny of phenomena with a view to secure further evidence on this subject.

DISCUSSION.

DR. LANE: Dr. Hubbard has expressed very clearly to us the formation of a certain type of amygdaloids along the surface of one flow, which in certain cases laps over, and we have unquestionably in the Kearsarge amygdaloid the process he has mentioned, where we have the same kind of trap, the same large crystals between two or more amygdaloids and under them, and in another place we have practically just the one type where the flow seems to be the thickest as at the Wolverine. I am not quite clear as to the connection between that and the other hypothesis at the Ojibway. Such a lava flow stops and there is an interval of time during which erosion may take place and we may have valleys in that surface and other flows may fill that surface which would be of an entirely different nature. That seems to be his idea in his description of the Ojibway, in which case only the lowest amygdaloid would belong to the Kearsarge flow and the trap between the other amygdaloids would be of a different, later character. I should like to be a little more clearly informed upon his idea in that respect.

DR. HUBBARD: I don't quite understand the reference to trap.

DR. LANE: The trap between the different veins in the Ojibway, if they were in a valley, they would be of a later flow than the main floor of the Kearsarge.

DR. HUBBARD: My idea is that these subsequent flows came from the same source and have the same chemical composition as the foot wall bed, separated from it and perhaps from each other by considerable time intervals; they came down these valleys and spread out at the end of the intervening ridge; they spread out over a wide area and that wide area was what would constitute the plane of weakness in this particular case, and the shearing movement would originate in the wide area and extend up to the surface, involving not only the amygdaloidal bed but the trap on either side as well. I am frank to say that we have not pursued our investigations far enough to substantiate all these points.

DR. LANE: The point is this, do you think the intervening trap between the different lodes is of some age quite a little later than the main foot trap?

DR. HUBBARD: No, it is a part of the main foot trap in which erosion took place before the other flows came down. (Question understood as referring to bar of trap that runs across shaft).

DR. LANE: I mean the trap between the second and third veins of this copper.

DR. HUBBARD: That and the two "veins" might be all one bed, so far as we know, and was certainly later than the great foot-wall trap.

DR. LANE: The other point which I would deprecate is the idea that the deposition of copper was from saturated solutions. I believe it is a case of chemical precipitation upon the rock which tends to make an acid solution alkaline. I won't enlarge on that because you will find it in my report.

MR. DENTON: Any further discussion on Dr. Hubbard's paper? The subject of the paper is of great interest and great

importance, and is becoming of increasing interest by the addition of various bits of information from different parts of the range, as illustrated by Dr. Hubbard's paper. If there is no further discussion on this paper, we will pass to one that is almost a continuation of it, in that it will treat of some results of measuring the movement in these amygdaloidal beds—so-called amygdaloidal beds, I presume we will have to use that term now. The paper has the title "Failures of the Rule of Following the Hanging in the Development of Lake Superior Copper Mines," by Professor Sperr.

FAILURES OF THE RULE OF FOLLOWING THE HANGING IN THE DEVELOPMENT OF LAKE SUPERIOR COPPER MINES.

BY F. W. SPERR, HOUGHTON, MICH.

The beds of the copper bearing series of rocks of the Lake Superior district are many thousands of feet in thickness. In these beds are found conformable belts of mineralization which extend over many square miles of area of varying thickness. The usual width or thickness of the lodes is something under fifteen feet, but occasionally diffusions of mineral are found as much as one hundred and fifty feet wide which apparently belong to the same belt. The distances between the different lodes vary from a few feet to several hundred, or more.

On the hanging-wall side, the mineralization nearly always cuts off sharply, while on the foot-wall side it goes out gradually into barren rock. Therefore it is obviously a good rule to follow the hanging-wall in the development of the mines. And it would scarcely be possible to lose a lode by a strict application of this rule, if the mineralization were continuous and if there were no faults or slips in the formation.

One of the chief difficulties in the way of locating the lode on the other side of a slip, is due to the circumstance that the hanging-wall of one lode is always the foot-wall of a higher lode. Therefore, if the fault has produced a downthrow on the other side of the slip or crossing, the rock presented at the end of the drift on the other side may possess the characteristics of foot-wall with a little copper in it, and in reality be the foot-wall of a higher lode instead.

If the distance between two mineralized belts or lodes happens to be about equal to the amount of displacement, the one

belt will be found in direct continuity with the other; the two will almost certainly be identified as one and the same on the opposite sides of the fault; and the continuation of each in opposite directions from the fault, will be lost. If a poor lode is thus made to take the place of a good lode, the change is popularly attributed to the influence of the crossing. (And is it not so?) The poorer lode may be followed a long distance at great and useless expense, until one of three things happens—first, the work may be stopped by discouragement or exhaustion of funds; second, by the accident of faulting, the drift may again encounter the better lode; or third, the poverty of the mistaken lode may compel cross-cutting until the better lode is found. There is only one chance in two that cross-cutting will be started in the right direction, and it is commonly observed that half the cross-cuts are started in the wrong direction.

The fracturing and faulting of the beds appear upon first acquaintance, to be most confusing and without law or order; but upon closer analysis, they appear to have occurred in separate sets or series, and that each different set occurred in a different period of time. The first set of fractures was probably developed by the cooling of the lava and was for the most part filled with vein matter, without producing much, if any, faulting. This set belongs to the period when the beds were in their original and nearly horizontal position.

The second set of fractures probably occurred contemporaneous with, or immediately following, the great break which allowed the country to the south-eastward to drop down. The down-throw of the south-east side probably caused the uplift on the northwest side. Anyway, we find a series of lesser faults running more or less parallel with the main fault, and gradually fading out with greater distance away from it. The strike lines of the fault planes resulting from these fractures, the strike lines of the uplifted beds, and the line of the great fault, are nearly parallel with each other; and the dip of the fault planes is in the opposite direction from the dip of

the beds, the former being toward the great fault and the latter away from it.

In the general settling of the formation to a state of equilibrium after the upheaval, there were other sets of fractures and displacements produced. How many sets may have occurred in any particular locality, has not been determined; neither has their chronological order. But displacements have been observed as illustrated in Fig. 1, which clearly indicate that there were definite sets of faults at different times. They are rather sharply distinguished from those of the two different periods already described. For convenience, as well as from necessity, we shall consider the time since the upheaval, as the third and last period in the history of the faulting. This period alone is represented in the illustrations given here as Figs. 1, 2, and 3. The faults appear to be altogether normal, but in many places they make up a highly complex system; and when this is complicated with the vein structure

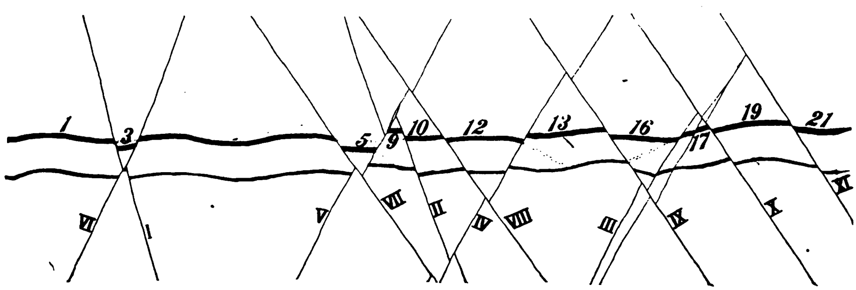


FIG. 1.

HORIZONTAL SECTION OF A FAULTED TERRITORY.

The Roman numerals I and II indicate the first series of faults here represented; III, IV, V and VI, the second series, and VII, VIII, IX, X and XI, the third series; all of the final period.

The Latin numerals indicate approximately the blocks having the same numbers on Figs 2 and 3.

The dotted lines indicate some of the courses which a drift may take in going from left to right in the process of developing a mine.


1
2
3
4
5
VII
6
7
VI
V
8
9
10
II
VIII
11
12
13
16
17

FIG. 2.


7
6
5
VII
12
15
16
18
20
21
XI
IX
X
19
10
VIII
II
9
13
17
14
11
8

FIG. 3.

of the first period, together with the faulting of the second or upheaval period, the complexity is greatly increased.

Fig. 1 represents a horizontal section through a territory faulted by three sets of faults of the third period, and containing two parallel lodes. The order of succession of the different sets becomes evident in this section. It shows that the set numbered I and II were displaced by the set numbered III, IV, V and VI; and this latter set was in turn displaced by the set numbered VII, VIII, IX, X, and XI. The last set showing no displacement, has not been crossed by any later faults within the area of the section.

Figs. 2 and 3 are views of a model made to illustrate the result of three sets of faults upon an individual lode. The dips of the fault planes and the order of the displacements are not shown in these views as they might be in a more accurately constructed model; but, taken together with Fig. 1, they serve our purpose. The different faults are designated by roman numerals in the order of their occurrence, the same as on Fig. 1; and the detached blocks of ground are designated by Latin numerals.

At first, two successive drops to the left occurred. All the country to the left of No. I dropped, and then all to the left of No. II went down. This concluded this set of fractures in the territory represented by the model. Then another set of fractures started with No. III, with successive drops off to the right as far as No. VI.

Some time after No. VI, a third set of fractures began with No. VII and ended with No. XI. All the territory to the left of No. VII fault participated alike in this fault, and the former relative positions of the different blocks of ground from 1 to 4 were not disturbed. No. 3 block did not participate in either No. I or No. VI drops and is, therefore, left standing above both No. 1 and No. 4 blocks, each of which took part in a single but different drop. No. I and No. VI drops having been of about equal magnitudes, the blocks 7 and 4 are left without displacement relative to each other,

showing a case where "on the other side of the slip the lode comes in its regular course without any heave," as the miner expresses the condition.

Before the long No. V drop occurred, 4, 8, 5 and 9 were one and the same block; and 6 and 10, and 7 and 12 were all one. After this fault (No. V), 8 and 9 went down below 4 and 5, and 10 and 12 went below 6 and 7. No. VII fault threw No. 4 below No. 5, and No. 8 below No. 9. The succeeding fault, No. VIII, threw No. 6 below No. 7, No. 10 below No. 12, and No. 11 below No. 13—No. IV having thrown Nos. 11 and 13 below Nos. 10 and 12. In the final adjustment, the distances of 7 above 12, 6 above 10, 5 above 9, and 4 above 8, are equal to each other and are measured by the No. V drop.

Nos. IV and IX faults, being of nearly equal magnitude, 12 and 16 blocks are left relatively undisturbed with respect to each other; 15 is left above 12 and 16 by the distance of either fault, and above 13 by the distance equal to the sum of both faults. In the same way 14 is below 16 by the distance equal to the sum of the drops of III, and IX; 13, 16, 18, and 20 are left above 14, 17, 19, and 21, respectively, by the amount of No. III displacement.

Let us now consider the difficulties to be encountered in driving drifts to locate these variously and more or less separated blocks of copper bearing rock.

On Figs. 2 and 3, three horizontal lines represent the drifts and three different levels of a mine. The upper one develops the blocks 1, 4, 12 and 16; passes over 10 and under 3, 5, 6, 15, 18, and 20. The middle one develops the same blocks of the lode (shoots of copper) as above, and finds a small bunch of 19. The level does not look quite as good as the one above. The horses of trap between 1 and 4, 4 and 12, and 12 and 16, are larger than above; and the new bunch of copper does not amount to much. The lower drift finds the same shoots of copper as the ones above. No. 19 has grown a little better, but the others have become smaller.

The level is indeed very poor, 1, 12, and 16 being almost entirely cut out by the trap.

It is not at all probable that the foot-wall blocks, 8, 9, 11, and 14, would be discovered; but rather that the large areas occupied by 8, 11 and 14 at greater depth, would be pronounced barren.

Where the drifts pass on the hanging wall side of Nos. 2, 8, 9, 10, 13, and 17, the popular opinion is that the trap has come in and cut out the copper. No displacement is recognized, because the lode is picked up again in each case and in its regular course on the other side of the horse of trap. But if the trap should be identified as foot-wall rock on account of being near the higher lode, the drift would be turned towards the hanging and into the wrong lode.

When the blocks 1 and 4 are stoped out, they are found completely enclosed by trap, except that they may break into each other where they come nearly together below block 3. Sometimes where the stopes are cut out by trap like that over 2, a little copper is in evidence near the foot but is not considered worth going after, because it is supposed that the trap has come down and pinched out the copper. And sometimes where the stopes are cut out by trap like that under No. 3, a little copper shows up near the hanging which is not considered worth drill and powder. No. 12 is another bunch of copper completely surrounded by trap. It may break into 16 and 16 may break into 19; 10, 13, 17, 18, 20 and 21, may perhaps be found by cross-cutting, or by accident, or may not be discovered at all.

Bad as the case is above made to appear, it might be worse if following the foot-wall were attempted, or if driving on a general strike-line were made the rule. Then if the shaft should happen to go down in No. 10, only a very small bunch of copper would be found at the higher level, and only the small bunches 2, 10, 13, and 17, at the middle and lower levels. The whole territory would be condemned as of little value. The upper level would be found practically all bar-

ren. the middle level would look a little better and courageous drifting would find No. 13, a small bunch in the trap above this level. The third level would find the conditions much improved, 10 and 13 coming almost together, which would probably encourage persistent drifting and lead to the discovery of 17 and possibly also of 2. The fourth level coming near the bottom of Figs. 2 and 3, would find No. 10 pinched out, 13 almost gone, but 2 and 17 improving. A zone of copper would be said to be making off to the right in going downward, as represented by 10, 13 and 17; and a second zone represented by No. 2 would be found bearing off in the same direction.

If the shaft should happen to go down in No. 7, drifts might go far in either direction at the higher level without finding the mineral. The lode might be altogether lost at greater depth. And should a cross-cut from the shaft some 300 feet below No. 7, discover No. 8, the drifts in either direction would find little copper.

Many other chances of hard luck in mining might be delineated on the small area covered by the illustrations; and in fact the conditions are not nearly so complex as many of those which are found in some of the mines.

The illustrations do not take into account the additional complications caused by the displacements of the second period These have been described as most numerous and greatest near the great fault; and therefore the detached blocks which are nearest to surface are the shortest and the most displaced. Shafts pointed on lines which cut the first few blocks, or bunches of copper as they are found in preliminary exploration, will go into the hanging-wall rock at greater depth; and the lodes are said to straighten up in depth. Stopes in going upward are suddenly cut off by trap, or the lode is found to "roll over" into the foot. Drifts are cut off in the same way when the strike line of the fault crosses that of the lode. A mine developed in this manner is said to be "bunchy," although, in fact, the mineralization may be continuous on the lode.

It is believed by some that a certain amount of leaching has occurred near the slips; but I have seen no certain evidence of such action. On the contrary, it has been observed that the copper is persistent up to one side of the slip, but disappears on the other side.

It is hoped that these illustrations may serve at least two purposes; first, to show how mines, which were at one time considered bunchy and unprofitable, in reality had continuously mineralized lodes and were made profitable after the true conditions had been determined; and the second, to suggest to mine surveyors and engineers the importance of the accurate location and identification of these faults or slips.

Those in charge of the underground operations have gained much by their long experience, and great improvements have been made in recent years over the conditions which prevailed in the early days and to which the large piles of poor rock at many of the mines give mute evidence. But it is also true that more efficient assistance should be given the mining captains by the mine surveyors in many of the mines.

DISCUSSION.

DR. HUBBARD: I would like to ask Professor Sperr if he has ever made any observations on scratches on shear planes grooves? Whether a study of those grooves and scratches will not determine the direction of faulting, that is the comparative movement of the two sides and the dip as well.

PROF. SPERR: Sometimes it does.

DR. HUBBARD: I think Prof. Carvill Lewis made some observations of that kind, and by using a sheet of paper and a soft pencil could trace scratches, and by noting the difference in width of the scratches, determine the direction of the movement.

ECONOMICAL LUBRICATION.

BY W. M. DAVIS*.

Lubrication, as we understand it in mechanics, is the application of or introduction of some substance that will cling to or flow between the surfaces of bearings and journals of engines and machinery and keep the metal surfaces from coming in direct contact, thus preventing excessive friction and consequent heating.

Lubricants can be divided into three general kinds of classes—fluids, plastic and solids.

To the first named class belong the various oils, to the second the greases and to the third such substances as graphite, talc or soapstone, mica, etc.

Where the speed is high and the pressure great, oils are, in nearly all cases the most satisfactory lubricants to use. They cling to the contact surfaces, forming an elastic coating to the metal, keeping them apart, absorb the frictional heat and carry it away. Another advantage they have, is, that they can be had in almost any desired grade or density, from the thin spindle oils to the heavy dense cylinder stocks, and they do not become rancid or gummy and contain no free acid.

As probably all of you know, in early days of engineering and mechanics, the oils used were all of animal or vegetable origin, such as lard oil, whale and sperm oil, rape seed or colza oil, olive oil, etc. These, while of good lubricating value had their faults or disadvantages. In the first place, they were expensive; being of organic origin they had a tendency to become rancid and the content of free acid increased on exposure to air, etc. In more recent years these have

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been almost entirely displaced by the petroleum or mineral oils. These are cheaper and, in many respects, better lubricants. Besides, they do not change their condition on exposure or in use, provided they are clean.

The greases are more suitable for use on slow moving machinery and where the pressure is not great. There are places, however, where the speed is comparatively high, but the pressure is light, where, if the proper grade or consistency be selected, a grease will often give excellent results. As a usual thing if grease be used indiscriminately on high speed machinery, such as textile machinery, there will be a noticeable increase in the friction load. But on slow speed machinery and mining and cement mill machinery it will lubricate just as well and there will be no apparent increase in friction load. In fact where the machinery is exposed, to much dust, grease, if of the proper grade, will prove more efficient than oil as it will act as a seal at the ends of the bearings and other openings to keep the dust out.

Greases may be divided into two classes, the lime and potash soaps or high melting point greases and the tallow base or low melting point greases.

The first are made by combining a small amount of fatty oil into a soap by means of lime water, caustic potash or other alkali, and mixing it with a large amount of petroleum oil, such as engine oil. Such greases may be made in any degree of density and will usually have a melting point of 140 to 180 degrees Fahrenheit.

The tallow base greases are composed of a large per cent of tallow combined with an alkali and brought to the desired density by means of vaseline, petrolatum or petroleum oils. Such greases, owing to their large content of tallow, are of low melting point, usually about 116 to 120 degrees Fahrenheit.

The high melting point greases usually require to be forced down between the journal surfaces by means of compression grease cups. The low melting point greases can

often be packed in the journal boxes or directly on the bearing, as is the case in top rolls of spinning and twisting frames, or on large open bearings of paper mill and other machinery and a low frictional heat will cause it to melt and change to an oil and lubricate the bearings.

The solid lubricants, such as graphite, soapstone, etc., have but a limited field of use, such as a filler or lubricant for fibrous piston rod packings, etc.

All the above is preliminary to a discussion of the subject matter of this paper, namely, Economical Lubrication.

Until recent years this subject has been given but little thought and attention by those in charge of the operation of machinery, the prevailing idea being that as the cost of lubricants was such a small matter, as compared with that of other items, such as fuel, labor, etc., that many went on the theory that, "oil was cheaper than babbit," and used plenty of it. But these days of keen competition and close attention to operating costs it has been found that it is possible to lubricate machinery efficiently and, at the same time, economically.

There are three essentials to be looked after in obtaining economical lubricating costs. First, to select such lubricants as will give good service as regards cool bearings, low friction loss, etc. Second, to obtain such lubricants at the lowest market price. Third, to see that they are used economically.

The selection of proper lubricants depends, of course, upon the class of machinery on which they are to be used. If, on light running and high speed machinery, such as in the spinning, twisting and in other departments of textile mills, the light bodied or more fluid oils will give best results. For slow speed machinery, the heavier bodied oils will be better. For use on slow speed engines where the oil is fed from cups a heavy bodied oil should be used. For high speed work and engines where continuous oiling systems are in use, a light bodied oil should be used. Cylinder oils have for their base what is known, in the oil trade, as cylinder stock, of which

there are two classes, the light colored or filtered stock and the dark or steam refined stock, the latter being almost universally used. This cylinder stock is high in flash test and viscosity, but of itself would not make a good steam cylinder lubricant under ordinary saturated steam conditions; being a petroleum product it has no affinity for moisture and will not stick to the wet cylinder and valve surfaces, so it is customary to compound this petroleum stock with a certain amount of fatty oil usually tallow oil or neatsfoot oil in order that it will emulsify with the steam and cling to the surfaces. Where the steam is fairly dry, three to six per cent. is usually sufficient. Where the plant is small and the lubricating cost is a small matter, the most satisfactory method of purchasing lubricants would be to buy of some reliable oil firm, but where the cost of lubrication runs into thousands of dollars, as is the case with many corporations, and it is desired to obtain suitable lubricants at the lowest market price, the best plan is to purchase on specifications, stating clearly just what is needed and awarding a contract for a year's supply to the lowest responsible bidder. Specification buying is fair both to the consumer and the dealer. Specifications are simply knowing what you want and stating so on paper.

The usual physical tests for gravity, flash test, cold test and viscosity, and in the case of cylinder oils, a chemical analysis for percentage of fatty oil, which are very easily made, will give any one who is familiar with lubricants a very close indication of the relative values of different oils. If one wishes to go further it is often possible to make actual service tests which will show absolutely any difference in the lubricating property of two or more oils.

The mechanical world owes much to the careful experiments into the theory of friction by such able men as General Morin, the late Prof. Robert H. Thurston and others and the information derived from such experiments has been of great value to engineers. The machines on which their tests were made, however, are not always available, but if it

is destined to make a series of service tests, take a bearing that is running under constant load and speed, such as the main bearing of a high speed engine. Place a thermometer in the bearing so that the bulb rests on the shaft, and maintain a constant feed of oil. Have another thermometer placed somewhere in the room near the bearing and out of drafts, so as to show the temperature of the room.

Commence the test when the engine is started, note the rise of temperature at frequent intervals: also that of the room, continue the test until the temperature of the bearing ceases to rise. Every bearing will in the course of a few hours reach a point where the heat is radiated as fast as it is generated. Deducting the temperature of the room from that of the bearing will give the temperature due to friction.

If the engine runs in the daytime only, the bearing will cool off during the night, then the next day repeat the test with another oil. This data may be plotted on diagram paper so as to show in a graphical manner the difference in rise of temperature with various lubricants.

While it is no doubt true that in experimental work it has been found that the co-efficient of friction often decreases with rise of temperature, yet in everyday practice it is safe to assume that of two oils the one that will keep the bearing the coolest, is the best lubricant, so in tests of this kind the oil shownig the least rise of temperature will be the better lubricant. Such tests can also be made in ring oiled bearings of motors, dynamos or shafting.

When making such tests it would be well, after finishing one test and before commencing another, to wash out the bearing with gasolene.

If it is desired to ascertain the lubricating value of two or more cylinder oils, take one oil and feed it at a given rate for a few days, then remove the cylinder head and wipe over the surface with a piece of soft white paper. If a good stain of oil is found it is evidence of good lubrication. If there is no stain of oil and a liberal amount has been used, it in-

dicates either that the steam is very wet, or that not enough fatty oil has been used in compounding the lubricant.

The same tests can be used to determine the least amount necessary to maintain good lubrication. By gradually reducing the oil feed and examining the surfaces from time to time the proper amount necessary to maintain good lubrication can be determined. Of course, where tests of this kind are to be made some means must be provided for easy removal of the cylinder heads.

Next to the question of buying, efficient and economical lubrication hinges on the methods of handling and applying the lubricant. Reliable appliances for feeding lubricants will save money. There are many ways by which every manufacturer can decrease their oil bill, notably by right methods of handling and using the lubricants. In many cases, too, by substituting one lubricant for another and perhaps cheaper variety which will do the work just as effectively, a great reduction in cost can be made. For instance, I have found cases where a high priced cylinder oil was used to lubricate general mill shafting and machinery when an ordinary machine oil would have answered better and at half the cost.

It is simply a waste of money to fit engines with a continuous oiling system unless all necessary precautions are taken to recover the oil used. As an example, I will say that several months ago I made an inspection of the mills of a corporation operating several large plants. At one mill where three large compound engines are in one engine room, 800 to 1,000 gallons of engine oil were used per month, not a drop was saved, what didn't go down the sewer was lost in wiping up. I advised that they be equipped with a continuous oiling system, but, at the same time, I urged very strongly in my report the importance of fitting the engines with pans and shields so as to prevent the loss of the oil. Instead of giving the contract to some responsible firm and holding them responsible for the result, they decided to have their own men do the work. The chief engineer, whose pre-

vious experience had been mostly on steam shovels, and who was but little used to modern steam plant refinements, undertook the work and applied all his skill as a pipe-fitter on putting up the piping, filters, etc., but made such a bad job of fitting the engines with pans and shields that the loss of oil, after completing the oiling system, was as great as before.

On the other hand, it is surprising how little new oil is really needed to make up the loss. As an instance I will say that at a certain plant in one of the suburbs of Boston, there are two vertical compound engines of several hundred horse power each, runnnig at 125 revolutions per minute about eleven hours per day. The oil flows in streams over the bearings, yet so well protected are they that a barrel or 50 gallons of oil lasts for months.

At another plant there are two single cylinder horizontal engines, about 24"x48", running at 100 revolutions per minute. Before being equipped with an oiling system these engines used about 200 gallons of oil per month at a cost of $50.00. After being equipped and means provided to prevent loss by leakage, etc., the monthly consumption was reduced to less than ten gallons. At the same time, a change was made in the kind of oil used, which cost considerably less money, with the result that these same two engines are running right along month after month on a cost for engine oil of less than $1.50 per month.

A rather mysterious loss of engine oil was traced to a peculiar cause. While making an inspection of the plant of a client in Illinois, I found by the records that from 300 to 400 gallons of engine oil were required per month to make up the loss on two compound engines. The engines were fitted with a continuous oiling system with excellent facilities for preventing loss, the cranks, eccentrics and cross heads being cased in to prevent the oil being thrown out. The oil was used freely on the cross head pins and guides, and under the piston and stuffing boxes and between the cylinder heads and the end of the bottom guide there was quite a large

cavity or depression in the casting. From the bottom of this was a drip pipe, but the engineer kept it closed, allowing the cavity to fill with oil, which, when the cross head went out, allowed the oil to flow along the bottom guide. The cross head shoe on the return stroke splashed the oil up against the hot cylinder head. As they were using steam in the high pressure cylinders at 150 pounds per square inch, there was sufficient heat to burn or vaporize the oil which was indicated by the cloud of smoke continually coming out of the cross head barrel. At my suggestion the engineer allowed this oil to drain into a tank with the result that this cause of loss was stopped.

Besides what was lost from this cause it was found that the oil soaked waste used in wiping up was sent to the boiler room to be burned. The installation of an oil and waste saving machine stopped this loss.

Another cause for loss often occurs in wiping up with waste. The oily waste then goes to the boiler room to be burned. In wiping up around engines it has been found by experiment that a pound of dry waste will, after being used and squeezed out by hand, weigh two pounds, or, as the writer found at one plant, there was a loss of one gallon of oil for every ten pounds of dry waste used. This in itself is quite an item. In most large plants it will pay to install some make of oil and waste saving machine, by means of which the oil is extracted and filtered and the waste washed, dried and used over again.

To give an idea of what this loss sometimes amounts to, the writer, while inspecting lubricating conditions at the mill of a client, found, on inquiry, that they were using waste at the rate of twenty-eight thousand pounds a year. Most of this waste was used for wiping up around the engines and machinery on which a great deal of oil was used. All of it was sent to the boiler room to be burned. As the waste was heavy with oil, it is safe to say that at the very least, two thousand gallons of oil, together with the waste, were lost per annum.

Receiving, storing and distributing lubricants are other important factors in economical lubrication. If the plant is a large one where several thousand gallons of oil are used per year, it will pay to provide storage capacity so that all the oils can be bought in tank car lots. In this way there will be a reduction in the price and also a saving in the amount of labor required to handle the oil.

But if it be a small plant where only a few barrels of each kind of oil are used per month, the oil should be kept in tanks so arranged that the barrels can be emptied into them by gravity. Care should be taken to see that the barrels drain out thoroughly. As the empty barrels are worth seventy-five cents to a dollar or more each, they are worth saving and should be kept in a cool place to prevent shrinking until enough have accumulated to make a carload. When sold, the amount received should be credited to the lubricant account.

It is customary in a large plant to have some one in charge of the oil house, to receive and store the lubricants, and issue or deliver them to the various engine rooms and departments, keeping a record of the amounts issued in a book or on a blank form provided for this purpose. This is quite important in a small plant and can be done without great effort or expense.

In one plant the various departments are provided with cans or small tanks of size sufficient to hold a few days' supply. The name of the rooms or departments to which the cans belong are stamped on strips of sheet brass soldered upon the cans. Leaky cans or cans with broken spouts tend to wastefulness. The repair man should periodically gather up all such cans and repair damages or fit new spouts before returning the cans.

No oils should be issued except on a requisition signed by the chief engineer, master mechanic or department foreman. In one small plant where the amount used does not warrant keeping a man especially to look after the lubricants, the oil house is placed in charge of the general storekeeper, and

opened only at certain times, half an hour or so in the morning, and the same time in the afternoon. The men come or send for their supply of oil at these times. At all other times the oil house is kept locked.

At the end of the month the amounts of lubricants issued should be totalled up and entered on a blank. The totals when multiplied by their price per gallon or pound will show the cost of each kind of lubricant used and also the amount used in each engine room and department and their cost. By comparing reports month by month it can be seen whether the cost is increasing or decreasing and in what department the differences have taken place.

By dividing the total cost by some unit of product or output such as tons, pounds, yards, kilowatt hours or whatever it may be, the cost of output may be determined and entered on the report sheet. There should also be a place to enter the amounts of oil purchased during the month, the number of empty barrels sold and the amount received for them. Thus a complete record of the lubricant cost can be kept on one sheet.

Considerable saving can often be effected by reducing to as few as possible the number of different kinds of lubricants. Many persons in charge of machinery have an idea that they must have some certain brand of oil, and that they will have all kinds of trouble if they should attempt to use anything else. In a factory the writer found that in one department the foreman insisted on having a special brand of oil, the price of which was such that it was costing $1.17 per machine per month. (There were more than one hundred machines in operation in his department). At all the other plants belonging to the same Company, this same make of machine was costing less than half that amount per machine per month for lubrication and just as good results obtained by use of a less expensive oil.

At the mills of a certain company where the writer had been engaged to reorganize the lubricating practice, it was

found that four different brands of cylinder oil, five brands of engine and machine oil and a dozen or more different kinds or grades of grease were in use. Some were fairly reasonable in price others inordinately high. In one engine room, the engineers had to have a certain kind of oil, in other engine rooms in the same building the engineers couldn't use this oil at all, but had to have something else. In the electric power house still another brand was called for, and the men in the pump house had to have something else. Each of the four brands had a different price. Analyses and practical tests showed that they were all of good quality, but no one better than another. A few plain common sense tests and demonstrations soon convinced the men of the fallacy of their notions, and after that one grade of cylinder oil and two grades of engine and machine oil answered every need. As to greases, it was soon shown that three kinds of grades would answer every requirement.

But the main thing in regard to economical use of lubricants is to train the hands to be careful in their use of the oil, a matter which requires constant attention on the part of the management, and where there are several plants it will often pay to have a good man take charge of this work.

PART II

Oil House at the Baltic Mine.

BY F. W. DENTON, PAINESDALE, MICH.

The concrete oil house of the Baltic Mine was designed and built to suit the lubricating practice developed after consultation with Mr. Davis.

The drawing explains the design with sufficient clearness.

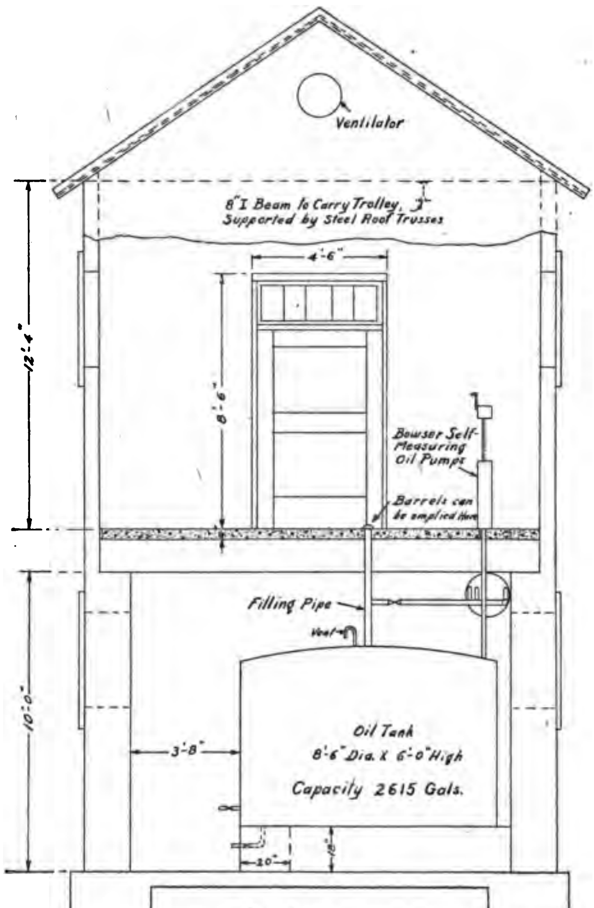
At present the oils are received in barrels, but as soon as storage is available at all of our plants we expect to obtain the cylinder and engine oils in tank cars, saving the expense of the barrels.

The cost of the Baltic oil house is divided as follows:

Building—		
Surface labor	$504.94	
Carpenter labor	131.24	
Machinist and blacksmith......	184.30	
Teaming	60.28	
	———	$ 880.76
Supplies—		
Cement	$297.77	

Sundries	277.70	
		575.47
Equipment—		$1,456.23
Tanks (not including kerosene tank)	$556.13	
Pumps	150.00	
		706.13
		$2,162.36

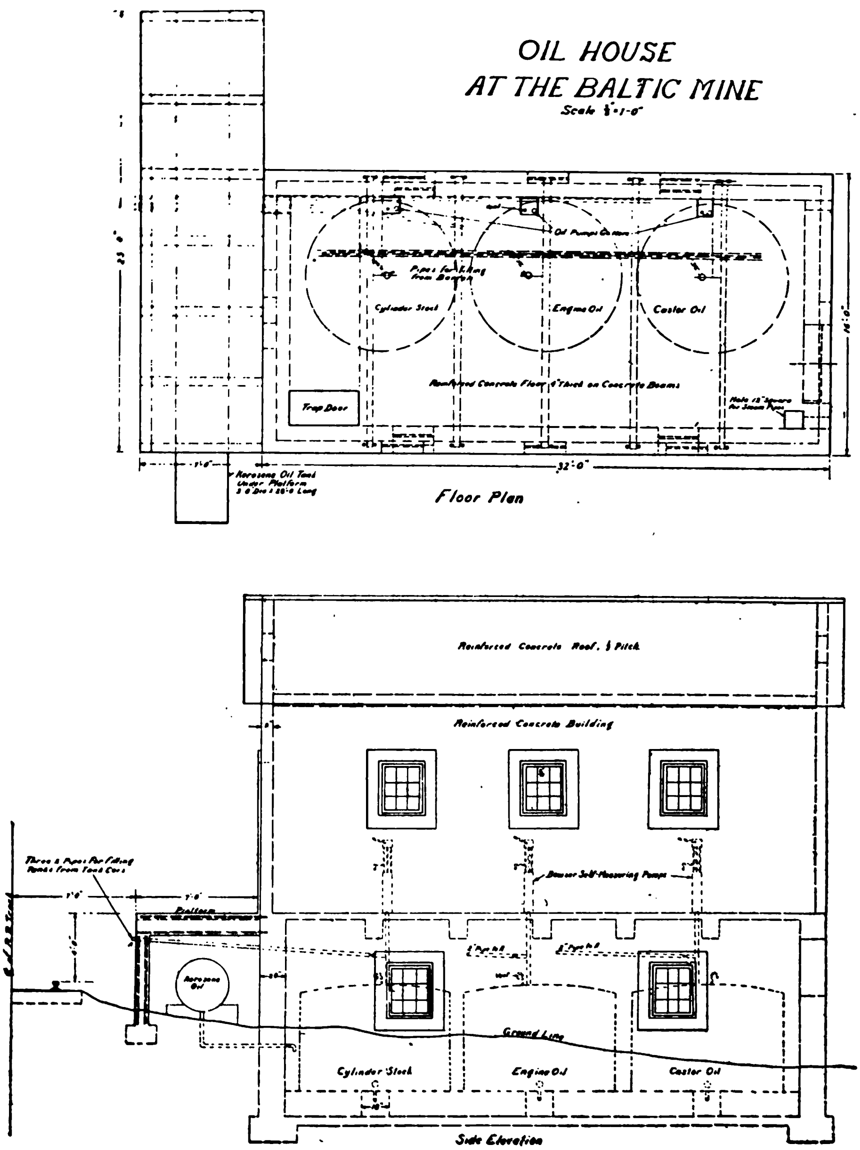
The total cost of lubrication for the three mines for the



Section

Oil House at the Baltic Mine

first six months of 1912, including oil used in shops and all greases, but not including lubrication expense at the mills was: Baltic Mine, $773.53; Trimountain, $576.51; Champion, $831.96.



METHOD OF RAISING, SINKING AND CONCRETING NO. 3 SHAFT, NEGAUNEE MINE.

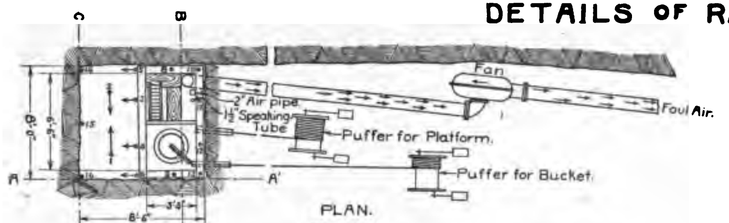
BY S. R. ELLIOTT, NEGAUNEE, MICH.

It was necessary to sink a shaft to replace two old ones which passed through the ore-body and were not permanent. It was essential to locate the new shaft at such a point that: first, it would not be endangered by future mining operations; second, that the necessary rock-drifting to reach the ore-body, and consequently the length of haul, would be reduced to a minimum; third, that sufficient stockpile-room could be provided. A location fulfilling these conditions closely was found and staked out on the surface. A test-pit was sunk through 68 feet of sand and hard pan to the ledge. As no water or quicksand was encountered it was known that serious trouble would not be experienced in sinking a shaft at this point. Driving the main drift from the bottom level of the old workings, which is at an elevation of 806 feet below the surface, was started and pushed as rapidly as possible to the location of the new shaft. When the drift was completed raising was commenced, the idea being to make a connection between the new drift and the bottom of the test-pit.

Raising vertically, in a single-lift, for a distance of 738 feet, had not been accomplished in the Lake Superior region previous to this time. Recently a similar raise has been completed at the Rolling Mill mine, the same method of timbering, ventilation, and hoisting the material being used. The following is a brief description of the raise:

Figure 1 shows the general dimensions and the layout at the level. The cribbed compartment was divided in the center by two-inch casing plank, forming two compartments,

DETAILS OF RAISE.



PLAN.
FIGURE 1.

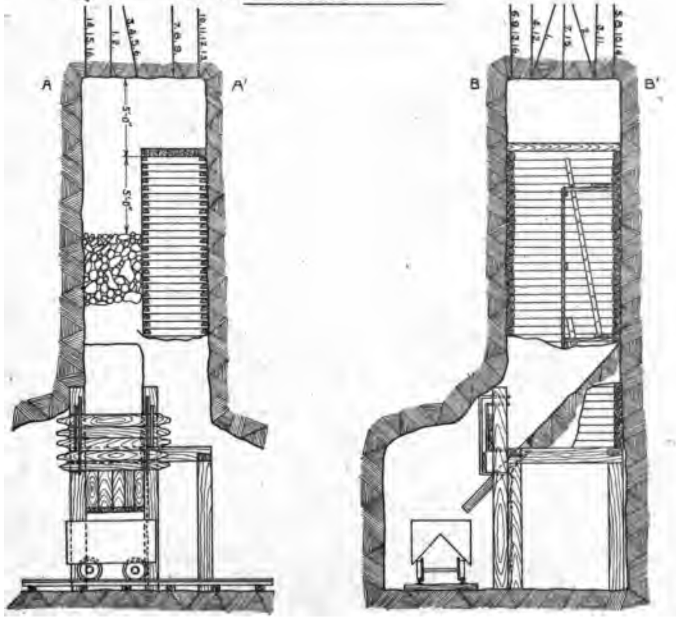


FIGURE 2.

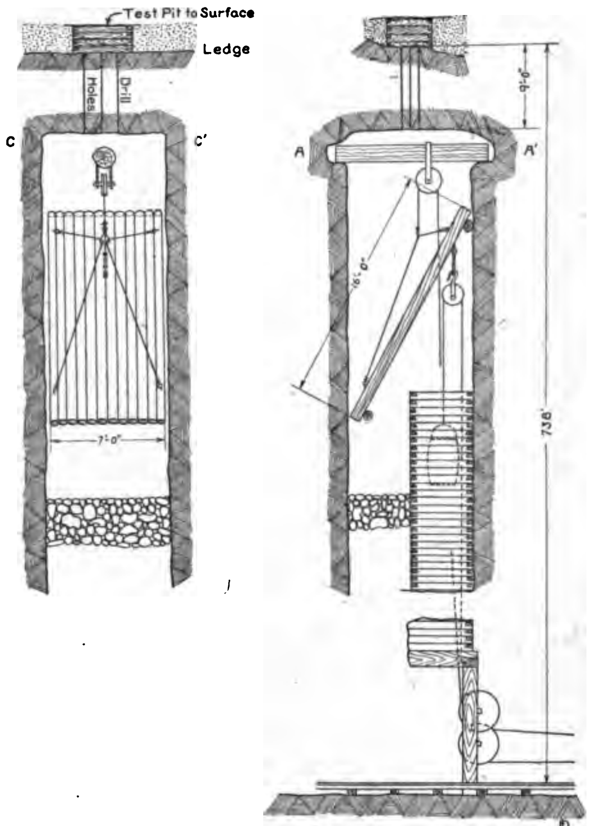


FIGURE 3.

one of sufficient size for the pipes and the ladder road, and the other for a small bucket to be used in hoisting tools and cribbing. Figure 2 shows the method of supporting the cribbing, and the arrangement of the chutes from which the rock was loaded into motor-cars.

Directly below the dirt compartment a bench of solid rock was left, running back on the same angle as the lip of the chute. This served as a permanent indestructible bottom. The cribbing was accurately framed from six to eight inch round tamarack timber. Each piece was faced on one side in order to maintain the same dimensions on the inside of the compartments. This greatly aided the miners in building up the cribbing plumb and made rapid hoisting practical, as there were no irregularities to strike the bucket. After the cribbing had been put in line it was necessary to wedge it very firmly in order to keep it from shifting.

DRILLING AND BLASTING.

The crew was composed of nine miners, three on each eight-hour shift. While drilling the cuts each miner used a Waugh No. 8 C stoper. It was found by experience that it was not advantageous to blast a deeper cut than five feet on account of the breaking or shifting of the cribbing. The best results were obtained by drilling the round as shown in Figs. 1 and 2. Before blasting, the cribbed compartments were covered with a six-inch sollar, one small hole being left in the ladder-road side to allow a man to pass through after blasting. The cut, consisting of sixteen holes, was blasted with fuse in three rounds as follows: 1st, holes 1 and 2; 2nd, 3 to 9; 3rd, 10 to 16. In order to make it practical to blast with fuse, at intervals of about 200 feet, small drifts about eight feet long were driven from the ladder compartment. Each drift was protected by a door. A connection was made with the air pipe to prevent the possible danger of the men being knocked out if, for any reason, the fan failed to work. It would have consumed too much time for the men to have gone to the bottom after the raise reached a great height.

Before blasting the crew would enter these small drifts and suffer no discomfiture even when the breast was not over 30 feet above them.

HOISTING.

A small puffer was located on the level and run by compressed air. After each cut was blasted and the loose rock in the back well trimmed, a stout pole was wedged across the raise as close to the back as possible. A ten-inch sheave was then hung from this pole so that the rope leading to the bucket was in the center of the hoisting compartment, the other end passing down one corner of the compartment to an angle sheave and then to the puffer. The signals were all given through a speaking tube which was simply a 1½ inch pipe extending from the puffer to the top set of cribbing. An oil barrel, reinforced in the bottom with wood and on the sides with strips of steel, was used in preference to an iron bucket. On account of its depth tools and cribbing could be hoisted with little danger of them falling out. The bulge in the center lessened the possibility of the bucket catching in the cribbing. All tools, supplies, and cribbing were hoisted to the breast, thus saving much time and labor.

VENTILATION.

On the level at a short distance from the raise a fan capable of exhausting 2,040 cubic feet per minute was set up. Ten-inch spiral-riveted pipe with flanged joints every ten feet extended from the fan up into the ladder compartment. The upper end of the pipe was kept as close to the last set of cribbing as possible. The discharge pipe, ten inches in diameter, was made of galvanized iron but was not perfectly tight. This fact caused a small amount of trouble. This pipe extended to an abandoned part of the mine. The foul air was discharged behind a tight door which prevented any of it reentering the main drift and contaminating the pure air which was needed in the breast of the raise. After blasting the fan was started, the gases being sucked into the pipe and the pure air from the main drift flowing naturally to the

breast. It was found to be a great advantage to run a piece of old hose above the sollar and discharge compressed air directly after blasting, as this hastened the removal of the gases. Before this was done there was a tendency for the air current to pass below the sollar. It was found that the gases were all removed within fifteen minutes, but as an extra precaution the men did not return to work for thirty minutes It was never found necessary to run the fan except while blasting the rounds.

PROGRESS.

On January 1st, 1909, the top of the raise was twenty feet above the rail in the drift. On July 19th, 1909, a hole was located in the back of the raise to strike the center of the test-pit, the size of which was two by three feet. The first attempt was successful, showng that the raise had been brought up with great accuracy. The total distance from the rail to the bottom of the test-pit is 738 feet. The last five feet was not blasted out until a later date. From the first of January up ro the last of April the ground was quite favorable for drilling, but the upper 250 feet was extremely hard, being sideritic chert. The greatest amount raised in any one month was in March, the footage being 142. No difficulty was experienced in completing the raise and if it had been necessary it could have been taken to a much greater height. As the raise increased in height the broken rock was allowed to accumulate, only enough being drawn out each day to keep it level with the top set of cribbing.

REMOVING CRIBBING.

Ordinarily in a raise which is to be stripped to a larger size, for a shaft, the cribbing is taken out during the process of sinking. Before blasting the men go down and remove and hoist to the surface the proper number of sets in order to make sufficient room for the rock which will be broken by the stripping cut. This operation is slow as the cribbing is difficult to remove. The miners have to stand on broken rock which is liable to settle without warning. The cribbing

is so damaged that it is useless for other work. The most serious disadvantage, however, is on account of the great danger of the raise hanging up. This almost invariably occurs. It is extremely difficult to break these jams and it is often necessary to hoist a large amount of rock to the surface. In a raise, over 700 feet in height, a jam occuring near the bottom would cause a great loss of time and expense. In order to avoid this jam and save the cribbing for other uses, it was decided to strip the entire raise of all cribbing. If this could be done the chances of the rock hanging up would be extremely small as the outside dimensions would then be about nine by nine feet. The method of removing the cribbing was as follows: A platform seven by sixteen feet was made out of eight-inch round tamarack poles bolted to cross-pieces as shown in Fig. 3. A wire rope was attached to each corner, their lengths being so adjusted that when the platform was suspended it would hang at an angle of 65° from the horizontal. Near the back of the raise hitches were cut and a strong timber put in place. From this timber a ten-inch sheave was hung. The platform was hoisted one piece at a time and bolted together in the raise. A five-eighth-inch wire rope passed over the sheave and was attached to the ropes from the four corners of the platform. The other end passed through a slot in the center of the platform, down one side of the hoisting compartment, to a small puffer on the level below The angle of the platform was so steep it would be practically impossible for any piece of rock, which might fall from the sides, to break through it. As the cribbing was removed and lowered to the level in the bucket, the platform was lowered and kept at the proper height above the heads of the men, protecting them absolutely from the fall of any ordinary mass of rock. The sheave for the bucket, which was used to lower the cribbing, was supported from the platform. In twenty-two working days the entire raise was stripped of all cribbing, pipe, and plank. No serious accident occurred during any part of the work in the raise.

The following is a detail cost of the raise:

	Labor.	Supplies.	Total.	Cost Per Ft.
Miners	$5,291.29		$ 5,291.29	$ 7.17
Fuse, powder, caps, etc.		$1,238.25	1,238.25	1.68
Drilling supplies and tools	726.79	331.23	1,058.02	1.41
Trammers	424.62		424.62	.58
Candles		110.27	110.27	.15
Cribbing	666.71	634.12	1,300.83	1.76
Timbering	690.23	190.00	880.23	1.20
Nails, iron and bolts		41.39	41.39	.06
Proportion electric haulage	199.80	50.14	249.94	.34
Proportion of air charge		260.00	260.00	.35
Shop labor and teaming		73.67	73.67	.10
Puffer	93 05	102.70	195.75	.27
Pipe and fittings	270.84	113.85	384.69	.52
Buckets and sheaves	74.60		74.60	.10
Total	$8,437.93	$3,145.62	$11,583.55	
Total cost per foot	11.43	4.26	15.69	$15.69
Cost of removing cribbing			1,067.59	1.45

PLAN OF SHAFT.

It was decided to make the shaft circular and to line the walls with one and one-half feet of concrete, inside diameter to be seventeen feet, which is sufficiently large to enclose a rectangular shaft of standard dimensions of ten feet ten inches by fourteen feet ten inches inside measurements. Fig. 4 shows the plan of the shaft and the general dimensions.

SINKING IN SAND.

After the raise had been blasted through to the test-pit, work was commenced on the surface to mill the sand into it in order to make a sufficiently large conical shaped hole to provide head room for installing the concrete equipment. The milling of this sand would also decrease the distance which it would be necessary to sink to the ledge. In a very short time this work was stopped. The sand, in falling for a distance of about 800 feet, packed so tightly in the bottom of the raise, that it prevented the water, which came through the rock, from reaching the level. When the sand became saturated it would rush out with great force. It was therefore not practical to handle it in this way. A derrick, with a ninety-foot boom, was set up on the surface and the remainder of the sand hoisted. At a point forty feet below the

original surface a hexagon-shaped wooden shaft, twenty-four feet inside diameter, was built. The sets were placed five feet apart and three-inch plank spiked to the outside. The alternate planks were cut five feet six inches and eleven feet in length (Fig. 6 D.) These served to tie the adjacent sets together, making the structure quite rigid and preventing it

PLAN OF NO 3 SHAFT NEGAUNEE MINE

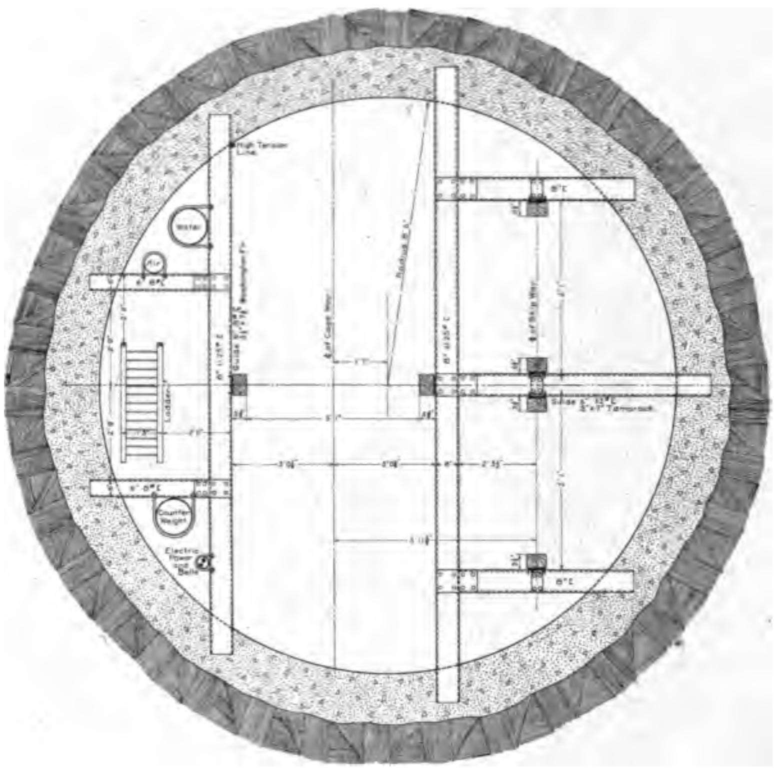


FIGURE 4.

from getting out of line. This shaft was allowed to drop slowly to the ledge as the material below was excavated.

CONCRETE EQUIPMENT.

A bed of gravel was found, at a point a short distance from the shaft, which was suitable for the making of concrete. From the conical shaped hole around the test-pit a cut

twelve feet deep was made to the gravel-pit and a track put in. Four feet below this track two pockets were built, the capacity of which was thirty-five yards. These pockets were provided with chutes from which the gravel could be drawn into a small car holding the exact amount for a charge. This car dumped into the hopper of a batch-mixer which was at a still lower elevation. Below the mixer the concrete was dumped into a kibble which rested on a small truck. This truck could be pushed to the center of the shaft, the kibble attached to the hoisting-rope and lowered, and the concrete dumped into the forms. The crew of five men on the surface could easily handle thirty-five yards of concrete on an eight-hour shift, or enough to complete ten feet of shaft. The gravel contained a small amount of loam and for this reason the proportion of cement used, about one to five, was quite high. The water was heated to boiling by exhaust steam, the tank being arranged with a gauge so that the proper amount was used with each charge. In the winter months the gravel was also heated by steam pipes in the bottom of the pockets. The cement house was located close to the edge of the conical shaped hole. A slide was built from one end of the house to the elevation of the hopper of the mixer. The bags of cement were thrown into this slide until it was full. As one bag was removed another would slide into its place. The general plan is shown in Fig. 6.

DRILLING AND BLASTING.

The crew in the shaft consisted of three miners on three-eight-hour shifts. One man on each shift acted as boss and was paid $3.25, the others received $2.90 per shift. On January 3rd, 1910, sinking in rock was started. This consisted of drilling a sufficient number of vertical holes, which depended upon the hardness of the material, and blasting them into the raise. After a careful investigation it was decided that the best progress could be obtained by using Murphy hand-sinkers. These machines drill rapidly, no time is lost in setting up, and they do not get out of order easily. While

the round is being drilled each man in the shaft uses a machine. The round usually consisted of eighteen holes six feet deep. In iron ore formation these holes could be completed in one eight-hour shift by three men. They were blast-

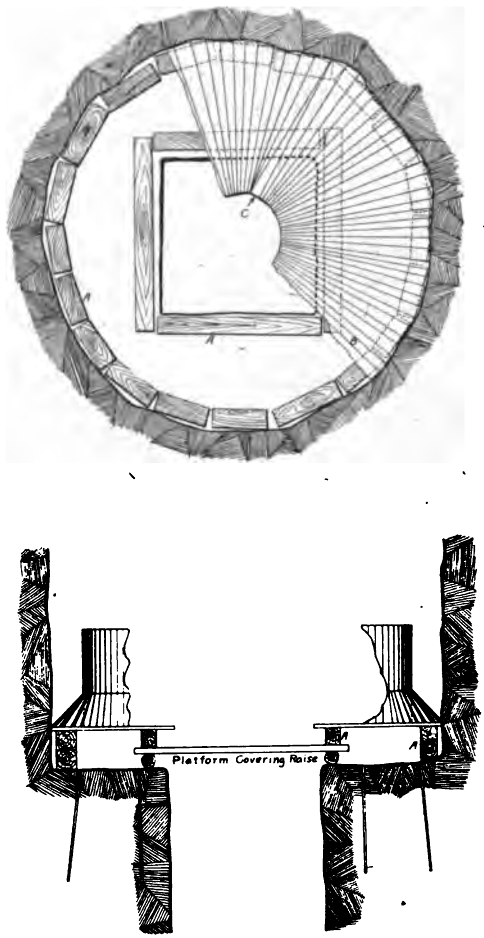


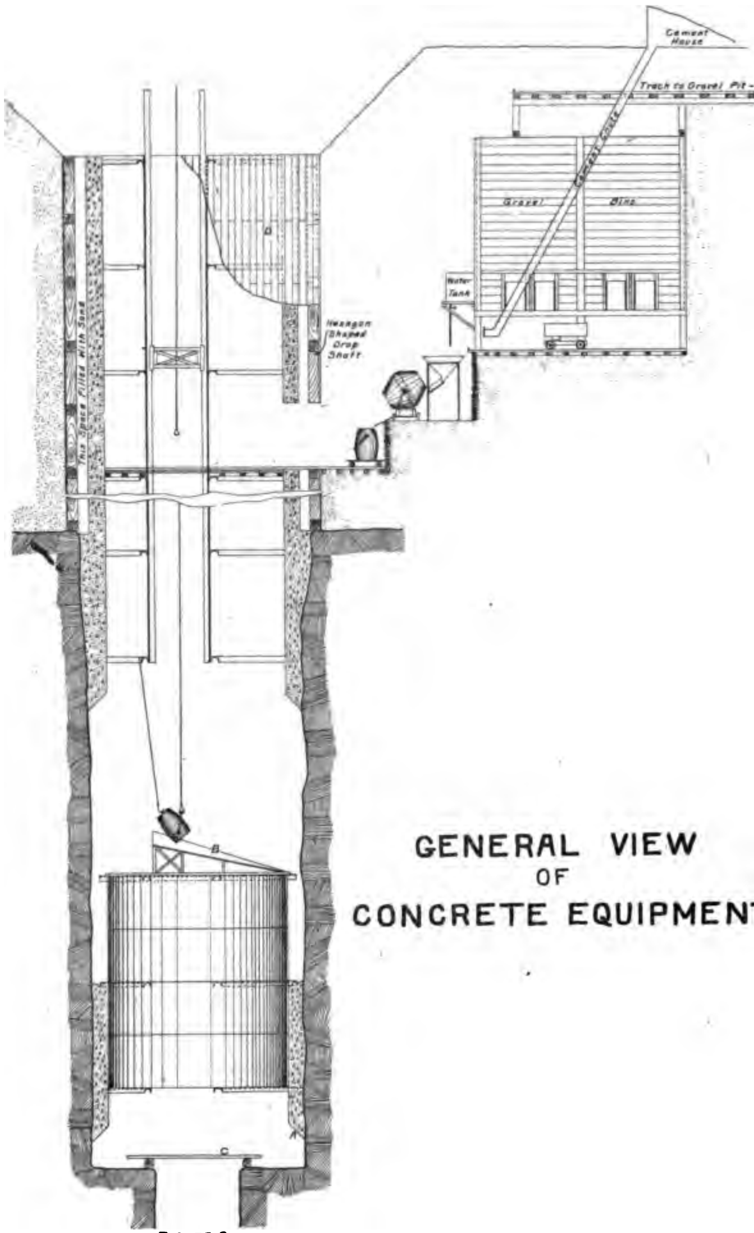
FIGURE 5.

ed in one operation by electricity, delay-action fuses being used. The general plan was to strip to as great a depth as practical. The sections were made multiples of ten feet and varied from twenty to as great as ninety feet according to

the hardness of the rock. After each blast the entire section was examined and carefully trimmed. If it could not be trimmed solid, stripping would be discontinued and preparations made to commence concreting. It is a great advantage to make the sections as long as possible as considerable time is lost in changing from one operation to another.

FORMS OF CONCRETING.

The concrete forms consist simply of rings which are five feet high and seventeen feet in diameter, made in four sections in order that they might be collapsible. They were properly braced on the inside to withstand the pressure of the concrete. When a certain section had been stripped the engineer would locate the elevation for the bottom of the first form. The procedure is to put in a number of short pieces of timber (Fig. 5 A) around the collar of the raise, and level them properly, to form a base for the floor upon which the forms will rest. The floor (Fig. 5 B) was made out of two-inch hardwood plank, seven feet long, sawed on a radius of ten feet, which is the radius of the outside diameter of the shaft. When the floor is laid on the blocking, the ends of the plank being shoved up against the irregular wall, it will completely cover the circumference. The small openings next to the rock are filled in with short pieces of wood to keep the concrete from running through. The plank being only seven feet long a hole about six feet in diameter is left in the center. In order to make it possible to remove the floor without cutting any of the pieces, one of the plank (Fig. 5 C) is laid down with the wide end towards the center. This can be easily pulled out and the others will then come freely. The first form lowered is a special one (Fig. 6 A) intended to cut off the lower face of the concrete at an angle of 45°. A regular five-foot section is then lowered and bolted together, being placed in position by dropping plumb lines from above. The special form fits snug up against this five-foot section, the upper circumference being one and one-half feet above the floor. The steel sets, which divide the shaft into compartments, are



GENERAL VIEW
OF
CONCRETE EQUIPMENT.

FIGURE 6.

lowered and placed accurately in position by lines dropped from above. They are then attached to the forms by simple clamps which keep them from shifting. Recesses were made in the top of two of the forms so that the top of the sets would be flush with the top of the forms, their ends extending about one foot into the walls as shown in Fig. 4. When the sets are once in position they serve as a means of building a platform on which the men can stand. The trough (Fig. 6 B) is next placed in position. This extends from the center of the shaft to the edge of the forms, the end next to the forms being about two feet lower than the part in the center of the shaft. This trough is light and the outer end can easily be moved around the circumference of the shaft. One-half yard of concrete is lowered in a kibble. A rope with a hook on the end is supported at a point high up in the shaft. When the kibble reaches the right point this hook is inserted in a ring in the bottom of it. The engineer continues to lower, and the kibble is inverted, the concrete running from the trough into the forms. After the first five feet is concreted, ten additional feet of forms are lowered and placed in position, a set again being put on top as already explained. In the part of the first section between the ledge and the surface special wooden forms were used as it would have required too much concrete to have filled the space between the steel forms and the hexagon shaped shaft. These forms were simply rings of strap-iron bent on a diameter of twenty feet, three-inch hardwood plank being wired to the inside of them.

As soon as fifteen feet of shaft is completed, or at the end of forty-eight hours, it is possible to remove the floor upon which the first form rested. The lower set of forms is also taken out and used in the work higher up. It is then practical to continue the stripping. The morning shift would concrete ten feet one day and on the following place in position ten feet of forms. By concreting on alternate days the crew on surface had sufficient time to fill the pockets with gravel. With only twenty feet of forms it was not possible

to average faster speed than five feet of completed shaft in twenty-four hours as it was necessary to give the concrete sufficient time to set. While concreting was being done on the morning shift the afternoon and night shifts would continue to strip and prepare the next section. The size of the crew was figured so that the two operations, concreting and stripping, would proceed at about the same speed.

COVERING RAISE.

During the process of stripping it was necessary to cover the raise to prevent accident by the men falling into it. A large strong platform (Fig. 6 C) was made, its size being about twelve by twelve feet. Wire ropes extended from the four corners to a hook in the center. This hook could be attached to the ring in the bottom of the kibble and the platform lowered until it rested on the solid ground completely covering the raise. Before blasting it was hoisted and suspended from the bottom steel sets in the shaft. This was found to be a rapid, practical, and cheap method of covering the raise.

MAKING JOINTS.

The object in cutting off the lower face of the concrete of the first section on an angle of 45° was to provide a means of obtaining a tight joint between two sections. When the lower section was completed the top of the form would then be in line with the thin edge of the concrete of the upper section. A special collar was then bolted to the form, its height being one and one-half feet, the upper edge being turned in towards the center of the shaft forming a lip. Rich concrete was rammed in between the ring and the old face. In the first operation the space was not completely filled. After the concrete had set the ring was removed and all leaks between the new and the old face corked tightly with oakum. The small triangular remaining space was next filled in with rich mortar. All joints between sections were made by this method and practically all of the water shut off. The shaft is not absolutely waterproof, a certain amount continually

seeping through the walls, but it is found that this seepage gradually decreases and, in time, it is thought that all of the small pores will become clogged.

WINZE.

It was necessary to sink the shaft to a depth of 155 feet below the ninth level in order to open up the tenth and to provide skip-pit room. At a point fifty-five feet north of the shaft a winze eight by eight feet, inside measurments, was sunk to a depth of 160 feet and drifts driven back to the shaft at elevations of 100 and 155 feet. From these two drifts a vertical raise was connected to the center of the shaft. The completion of this raise made it unnecessary to provide means on the surface at the new shaft for handling rock, as that which came from the part below the ninth level could be hoisted in the winze and trammed back to the old shaft.

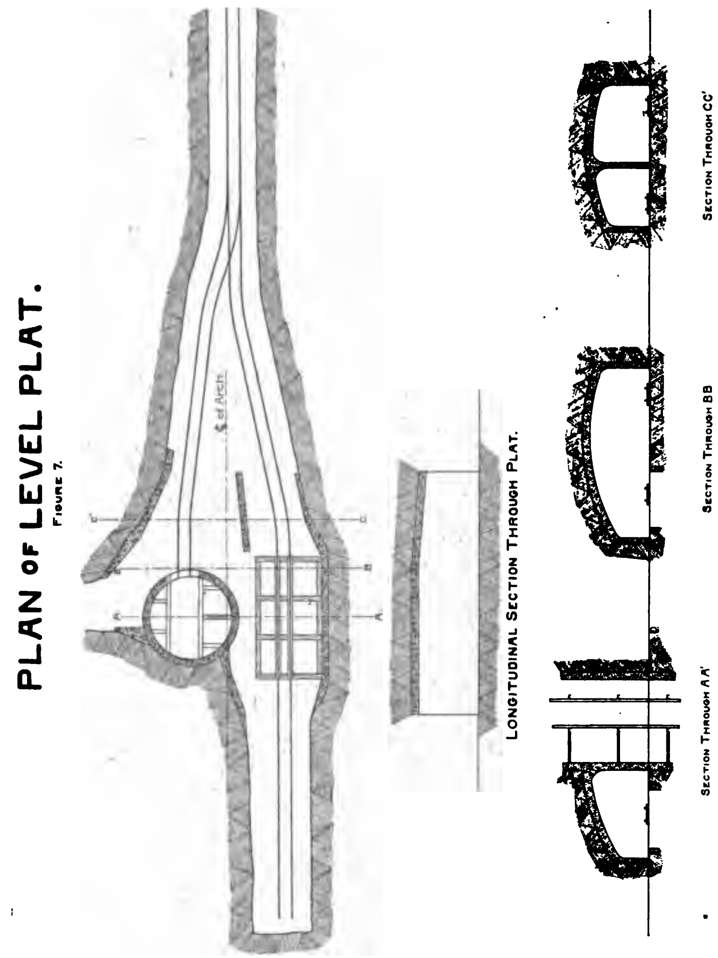
PROGRESS.

On January 3rd, 1910, sinking in rock at the ledge was commenced. On February 1st, 1911, the total depth of completed shaft was 767 feet, or an average speed of sixty-four feet per month. Below this depth the progress was slow on account of the large amount of work on the ninth and tenth levels in cutting and concreting extensive plats and storage pockets. Our principal aim was to balance the crew in such a way so as to complete the work at the smallest cost rather than to obtain greater speed in sinking.

EXCAVATING FOR PLAT AND POCKETS.

After the shaft had been completed to a point twenty feet above the rail of the ninth level the stripping was continued to the ninth level. The excavation was then made for the plat. After this was finished work was started in cutting out the rock for the storage pockets. On account of the span and height of the back, over the pockets, this was dangerous and slow, and it was finally necessary to put in temporary timbers in order to protect the men while excavating for the pockets. On the tenth level a different plan was followed,

which was safer, making it possible to finish the work in a shorter time. As soon as the excavation for the plat was completed it was concreted before any attempt was made to make room for the pockets. This protected the men from



any possible fall of ground, while excavating for the latter, and under this condition the work could be done more rapidly.

CONCRETING PLATS.

In constructing the plats, (Fig. 7) the side-walls were first put in. The concrete was dumped at an elevation of

twenty feet above the rail into a trough which carried it from the center of the shaft to the forms. After the side-walls were completed the forms for ten feet of arch were set up. These consisted of skeleton arches made in short sections and bolted together. On top of these arches, which were placed at intervals of five feet, two-inch hardwood plank was laid. In order to prevent the concrete from running between the cracks, and to make the surface smooth, it was covered with heavy tar paper or rubberoid. The arched roof of the plats are in the form of a section of a cylinder. As the skeleton forms were made in short sections the increase in width could be obtained by bolting together the proper number of sections. The concrete was dumped into a trough at a point twenty feet above the rail and in the center of the shaft. This trough was supported on light timbers and carried close to the back of the excavation and extending to the forms. The arch was so planned that it sloped down from the center of the shaft, the drop being one foot in ten. With a little shoveling concrete will run on this grade. In the highest part of the section it was necessary to ram in by hand the small amount to completely fill the space. At intervals of three feet holes were drilled in the back into which split I-bolts were driven and from these I-bolts a network of wire rope was attached. Where the side-walls joined to the arch short pieces of scrap iron was used as extra re-inforcing. To make the concrete waterproof two per cent of silicate of alumina was added to the weight of the water used. The results obtained with this material were excellent.

STORAGE POCKETS.

In order to be able to handle rapidly two grades of ore and rock, three-compartment storage pockets were put in. The general plan and dimensions of these are shown in Figs. 8, 9 and 10. From the center compartment it is possible to load into either skip. If the grade of the larger tonnage is handled through this compartment, and a little care is used to keep a reserve in the pocket, it will never be necessary

to run either skip up empty in order to continue to hoist from either of the other compartments. This may occur with a two-compartment pocket if the grades of ore hoisted are of different proportions. With the exception of the front, which is made of wood, the entire pocket is concrete. The sides are lined with plates, those in the lower part being one-half inch thick, while in the upper they are only one-quarter inch. The long dimensions of these plates extend horizontally instead of vertically, the object being to gradually decrease the thickness from the bottom, where there is the most wear, to the top where they would last for a great length of time. At intervals of two feet six inches on the sides there are five- by seven-inch timbers, (Fig. 10 A) to which the plates are bolted. These plates serve as forms behind which the concrete is dumped. As a rule, plates in pockets wear rapidly, due to the fact that they are not on a perfectly solid foundation. They buckle and in a short time ore gets behind them and they are shoved out of position. Plates in a concrete pocket are expected to last until they are completely worn out. Some way had to be provided for the replacing of the plates when worn out. As the concrete completely fills the space back of the plates it would be impossible to reach the nuts of the bolts, so the countersunk heads of the bolts were made like the head of a screw. Before dumping in the concrete the threads were well covered with grease to keep it from sticking, and after it had set for a sufficient length of time, so that there was no chance of the nuts turning, every bolt was loosened with a large screw driver and tightened again. When the plates wear out the bolts can be removed and other plates put in position, the same nuts being used as many times as necessary as they are firmly imbedded in the concrete. The bottom of the pockets are made out of two thicknesses of three-inch hardwood plank spiked to five- by seven-inch timbers, placed at intervals of two feet six inches. Behind these planks concrete was also dumped. When the upper planks wear out they will be removed and replaced by others.

STORAGE AND MEASURING POCKETS.
FRONT ELEVATION

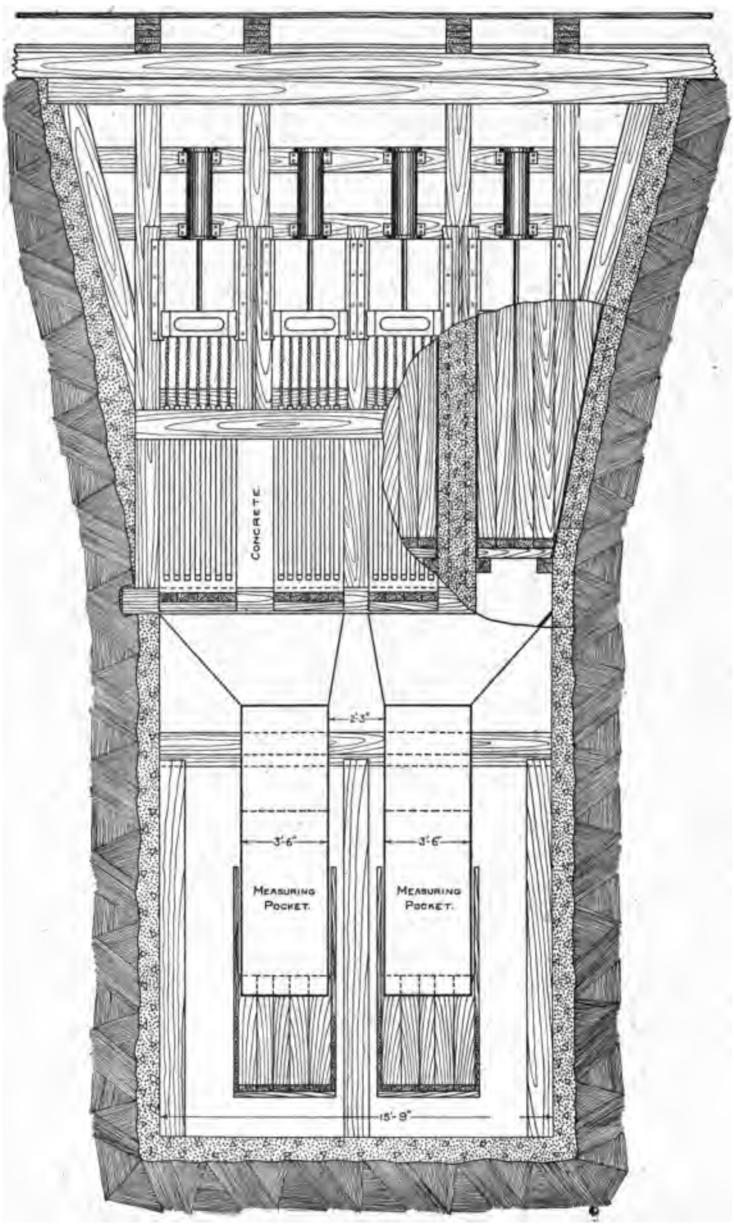


FIGURE 8.

STORAGE AND MEASURING POCKETS.

SIDE ELEVATION

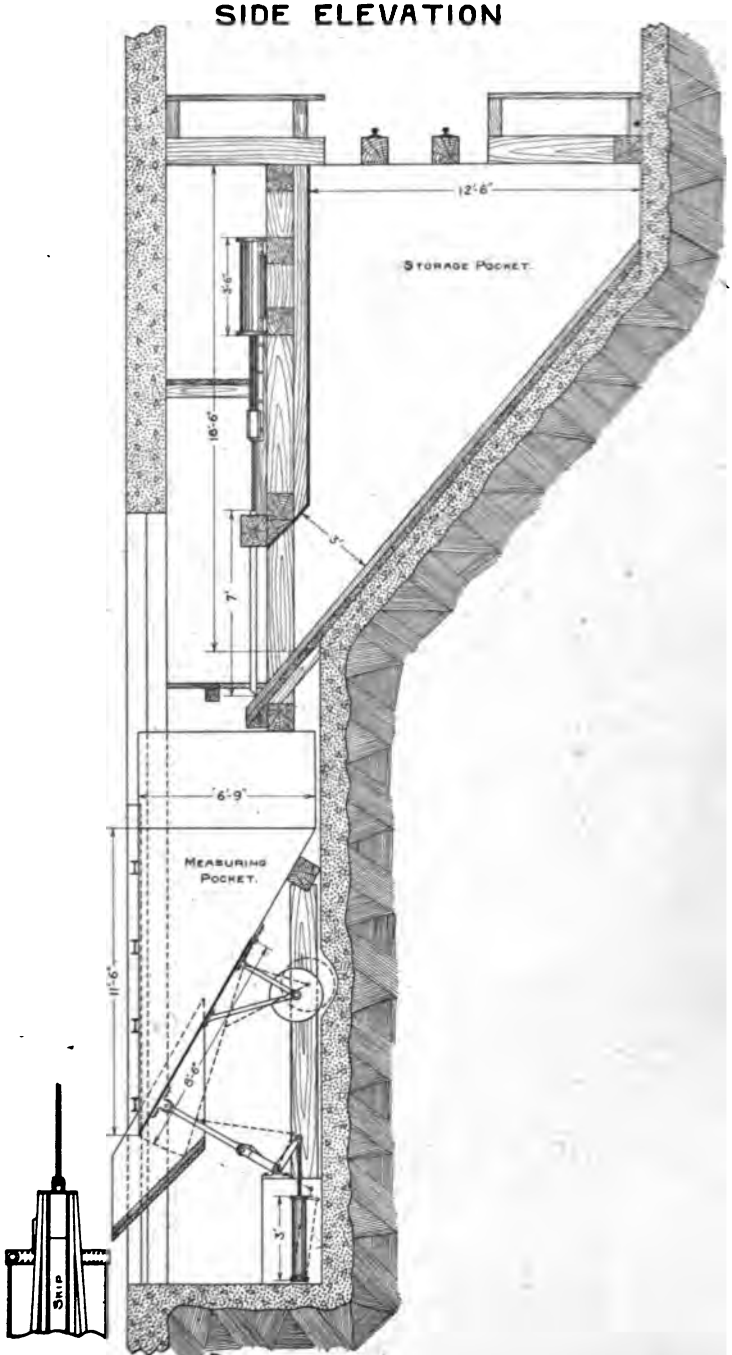


FIGURE 9.

MEASURING POCKETS.

Below the storage pockets there are two measuring pockets which hold a skip load. While the skips are in motion the operator has time to draw the ore from the storage pockets and fill a measuring pocket. As soon as the empty skip

STORAGE AND MEASURING POCKETS.

PLAN

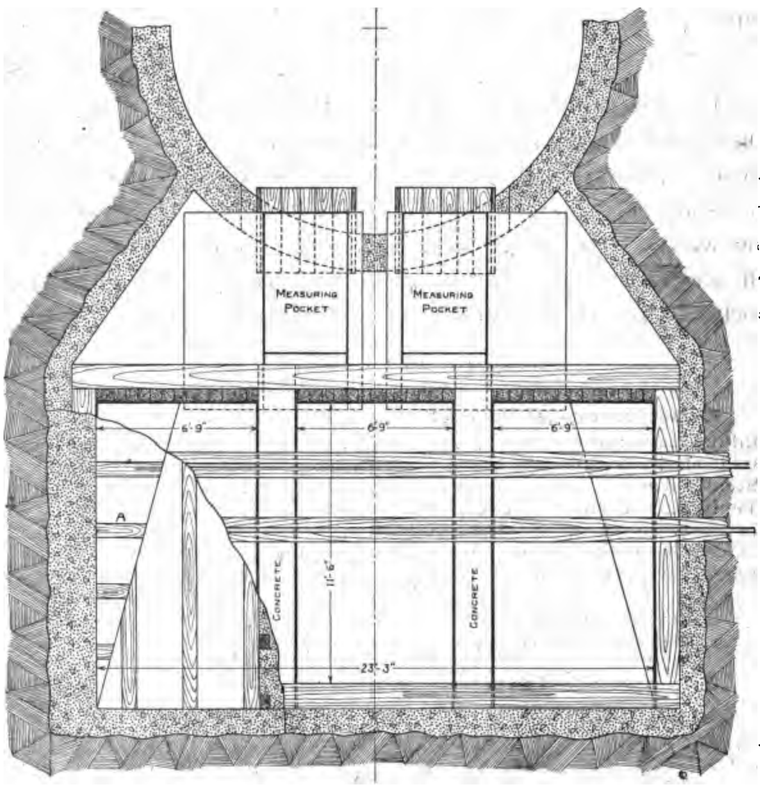


FIGURE 10.

is in position it can be immediately filled by simply tripping a lever. One great disadvantage in the common style of measuring pocket is that the door opens towards the skips. If, for any reason, the operator fails to close the door before giving the signal to hoist, it will be torn out by the skip,

causing great delay and damage. In order to avoid this possibility, and to make the operation as near "fool-proof" as possible, the pockets were designed with the doors opening away from the shaft. The ore falls on a slide from which it runs into the skips. The doors are made of two thicknesses of one-half inch plate strongly braced by angle iron. They are counterweighted, closed, and opened by small air cylinders. These have been found to work most satisfactorily and very rapid loading is possible.

COST.

The following statement shows the detail cost of the completed shaft. The part below the ninth level is considerably more expensive than that above due to the fact that it was necessary to transfer all of the rock, it being hoisted through the winze. The progress in this part of the shaft was slow on account of the large excavations necessary for plats and pockets on both the ninth and tenth levels.

DETAIL COST OF SHAFT.

	Amount	Cost per foot
Surface to Ledge—48 Feet.		
Sinking in sand	$ 3,638.76	$ 75.81
Steel shaft frames	493.56	10.28
†Steel forms	56.00	1.17
†Temporary surface structure and equipment	350.74	7.30
Cost of concrete	919.45	19.16
Estimated charge for compressed air	48.00	1.00
Temporary steam line in shaft	392.25	8.17
Total cost per foot	$ 5,898.76	$122.89
Less salvage		.12
Net total cost		$122.77
Ledge to 9th Level—738 Feet.		
*Cost of raising	$13,789.19	$ 18.68
Cost of stripping	12,787.20	17.33
Steel shaft frames	7,420.27	10.06
†Steel forms	927.87	1.26
†Temporary surface structure and equipment	4,975.78	6.74
Cost of concrete	12,710.06	17.22
Estimated charge for compressed air	738.00	1.00
Total cost per foot	$53,348.37	$ 72.29
Less salvage		.12
Net total cost		$ 72.17

9th Level to Skip Pit—158 Feet.

Cost of Winze, drift and raise	$ 3,861.52	$ 24.44
Cost of stripping	5,696.16	36.04
Steel shaft frames	3,468.08	21.95
†Steel forms	189.00	1.20
†Temporary surface structure and equipment	1,571.73	9.95
Cost of concrete	2,910.34	18.42
Estimated charge for compressed air	158.00	1.00
Total cost per foot	$17,854.83	$113.00
Less salvage		.12
Net total cost		$112.88
Grand total	$77,101.96	$ 81.68
Less salvage	112.58	.12
Net grand total, 944 feet	$76,989.38	$ 81.56

*Cost includes ventilating plant.

Cost per yard for concrete.......7.64

Average thickness of walls.......16.2 in.

†Cost pro-rated.

The method of construction at the Negaunee No. 3 shaft was similar to that used by Mr. Kelly at the Brier Hill shaft*. We are greatly indebted to him for valuable information and access to his work. Credit is also due to Mr. C. J. Stakel, engineer in charge of the work during the period of sinking to the ninth level.

*The Brier Hill Concrete-lined shaft, by William Kelly, Vulcan Mich., Vol XIV, 1909, page 140.

ROCK HOUSE PRACTICE OF THE COPPER RANGE CONSOLIDATED COMPANY.

H. T. MERCER, PAINESDALE, MICH.

The methods of handling copper rock in preparing it for the stamp mill have undergone some notable changes in the past few years, and it is the object of this paper to describe briefly some of the methods, past and present, peculiar to the Copper Country. As one locality is similar to another in many respects, I shall confine myself principally to the Copper Range Consolidated Company's mines.

The ends to be attained in a rock house are: The separation of the waste rock and copper rock, segregation of the mass copper, and the reduction of the stamp rock to a size suitable for the mill. The first step on reaching surface is the dumping of the rock, and it might be well to consider the various types of dump used.

In the early life of our mines the rock was dumped from skips, holding about two and one-half tons, by means of the old style dump, more or less common, throughout the country. In this dump the main stringer was cut away at the side sufficiently to allow the front wheel of the skip to drop through on a curved rail, toward the foot, or downward, the rear wheel continuing in the plane of the dip on an outer rail or strap, the tread of the rear wheel being wider than that of the front. Later, to accommodate a longer skip of about four and one-half tons capacity, this side, or outer rail, was curved outward, or towards the hanging. This was to throw the rear of the skip out, so that the nose would not project too far into the dump. Later, when a six-ton skip was introduced, a dump was developed by Mr. John Angove, Carpenter Fore-

man at the Champion Mine, in which the outer or back rail was raised still higher, and the main stringer continued through the dump without cutting, the front wheels remaining on the main track. In this dump the whole operation of dumping is performed by the rear wheels and curved outer rail. The nose of the skip is prevented from falling back by 12-inch iron rollers with a four-inch face, placed between the main stringers in the proper position, one- by four-inch straps on skip bottom engage these rollers. By inserting a hinge in the tail end of the curve which intercepts the rear wheels, it may be raised, allowing the skip to pass through without dumping. It is therefore possible to place this dump at any point in the shaft and use it as a poor rock dump, without weakening the main stringers in any way. Plate I shows this dump in outline, with general dimensions.

In our old rock houses, the rock, on leaving the skip dropped on an iron door about five feet long, set at an angle of about 30 degrees, below which, set at the same angle, were the screen bars, or grizzlies. The iron door was hinged at the lower end and could be raised to allow for the dumping of waste rock into a separate bin or car, underneath the skip track.

The grizzlies were first constructed of four-inch round iron bars, about 12 inches long, set four inches apart. These were later discarded in favor of three- by ten-inch hardwood bars capped with one- by four-inch iron straps. The latter were cheaper and easier to replace than the round bars. At the lower end of the screen was a four-foot drop to the floor, and at this point in the floor several bars, or rails, were inserted, set four inches apart, at right angles to the grizzlies, forming a floor screen. Any fine dirt that had failed to drop through the grizzlies could be shoveled through this floor screen to the storage bins below.

The crushers used at that time were of the jaw pattern, 18 x24 inch in size, and ran at a speed of 85 to 90 r. p. m. There were two of these crushers, one in front of each grizzly, and

about 12 feet away, the floor between the crusher and the grizzly being paved with heavy plates. This long space was to allow for sorting and picking of mass, but necessitated a great deal of shoveling to get the rock to the crushers. The mass copper was picked out and taken, by hand, across the floor to a drop hammer, where the surplus rock was pounded off. It was then thrown through a chute to the ground, where it was collected and teamed to a central point, to be loaded onto a car and shipped to the smelter. Pieces of rock too large for the crusher were also taken to the hammer and broken. The finer pieces resulting were dropped through an opening in the floor to the bins and the large pieces taken back to the crushers. Beneath the crushers were the large storage bins, from which the rock was loaded into cars by means of the ordinary apron chutes operated by separate hand levers. Any waste sorted on the crusher floor was transferred by hand or barrow to the poor rock car, which ran on a track a little below the level of the main floor. Most of the waste, however, was hoisted separately, dumped directly into the poor rock car and trammed out onto the waste rock pile by the rock house crew.

Such, briefly, were the methods in use ten or twelve years ago. Let us turn now to the newer method and see what has been accomplished.

In the first place, we have done away entirely with the grizzly. On leaving the skip, the rock drops directly into a large bin, capable of holding about 20 or 25 skip loads. (See Plate I). In the front side of this bin are two sliding doors, one in front of each dump, each operated by an eight-inch air lift. These doors are five feet wide and five feet high and are lined with two-inch cast iron plates. The front of the bins above the door is also lined with iron to protect the wall from the impact of the rock. The bottom of the bin is about two feet below the bottom of the doors and this space is filled with waste rock until it takes its natural slope to the door. No paving is used except near the door, where a heavy iron plate

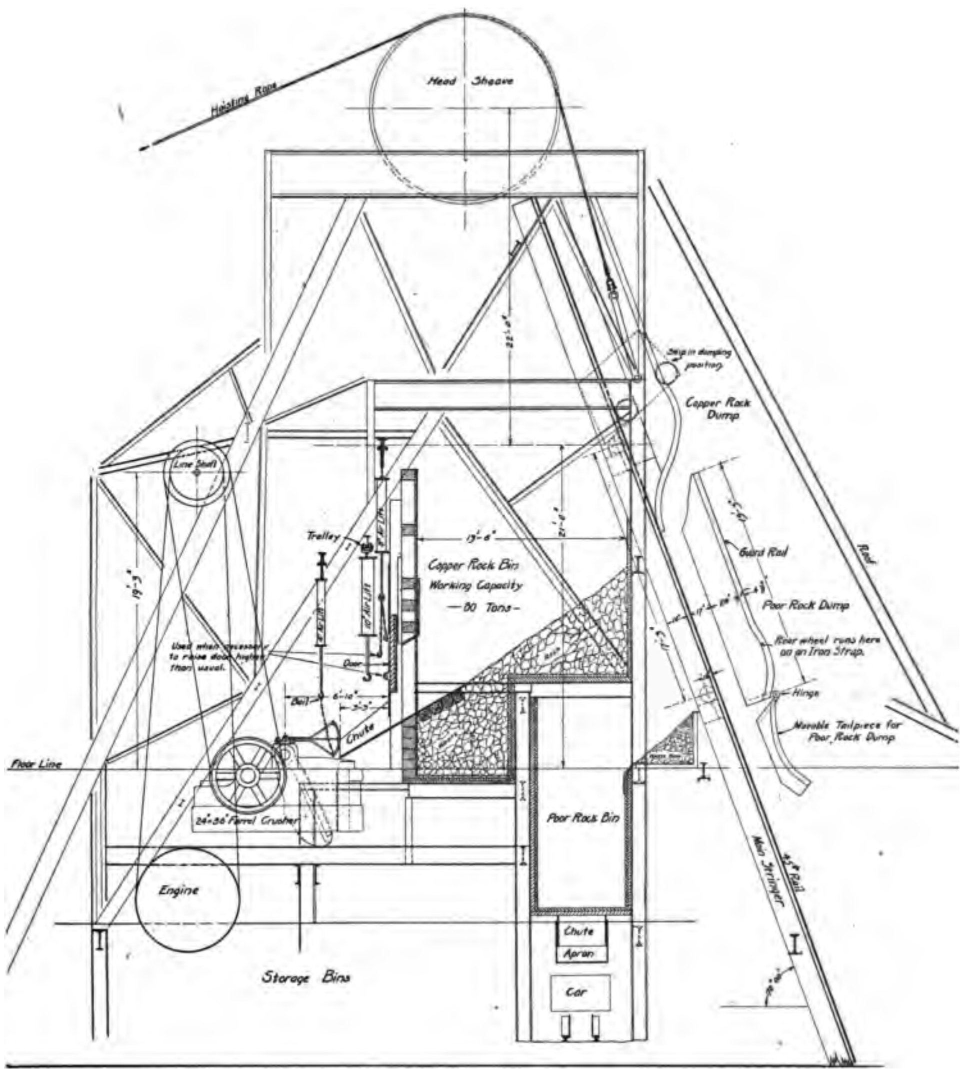


PLATE I.—Sectional Elevation of Rock House of the Champion Copper Co.

projects into the bin at an angle of 30 degrees, to help in starting the rock through the door. Outside of the bin, in front of each door, is an apron or chute about four feet long, set at 30 degrees. The lower end of the chute rests directly on the jaw of the crushers, which is a Farrell type 24x36 inch in size, run at a speed of 155- to 185- r. p. m. They are designed for as high as 300-r. p. m. There are two of these crushers, one in front of each door. Each crusher has a capacity of from 500 to 1,000 tons daily. Fastened to a shaft on the top of the crusher, by means of arms, is a cradle or gate, which drops into the lower end of the chute. This gate is operated by a four-inch air lift. The operation of feeding the rock to the crusher is as follows:

When the bin door is raised the rock runs down into the chute and is stopped at the crusher jaw by the cradle gate. The bin door is then shut and any mass or large boulders that may have come down are sorted out. Then the gate is raised and the rock feeds directly into the crusher with little or no handling. A trolley, carrying a ten-inch air lift, runs directly over the chutes in front of the crushers, and by this means the mass and large rocks are quickly picked up and transported to the drop hammer, which is set at one end of the building, and operated by a small electric hoist. On leaving the hammer the mass is pushed through a door in the side of the building and drops straight down to a platform alongside of the railroad track. This platform is built of log cribbing filled with waste rock, the top being at the same elevation above the track as an ordinary flat car. Hinged on the side of the rock house above the platform, in such a position that it swings out over the car, is a simple jib crane, built of an eight-inch I-beam, ten feet long. On this beam runs a trolley to which a chain block can be attached. By this means the mass copper is easily loaded onto the car. The larger masses are picked up with a chain, and an iron pan swinging on chains, is used for the smaller pieces. No sorting of poor rock is done in the rock house. All waste hoisted is dumped

into a separate bin by means of a lower hinged dump such as has already been described. From this bin it is loaded into a car, trammed out on a trestle and dumped through a raise into the old stopes.

At several of our shafts where the poor rock trestle is of sufficient height bins have been built near the end of the trestle, about six feet back from the hole, or raise. Over these bins just under the car track, is a screen or grizzly, about 10 feet long, with bars 1½ inches apart, set at an angle of about 35 degrees. When the car is dumped, the waste rock slides over the grizzly. The larger pieces go over into the raise and the fines drop through the screen into the bin, where they become available for road material and concrete. When no rock is being drawn from the bin it simply fills up to the screen, and everything goes over into the mine. At one of our rock houses, where the old screen bars are still in use, and the crushers are set 12 feet away from the screens, large feed pans or "tilting" pans have been installed, similar to those in use at some of the Calumet & Hecla rock houses. The pans are hinged at the crusher end, and extend from the crusher to the grizzly. At the screen end they are attached by means of a bail and chains to a large air lift, by which they are operated. On being raised they stand at an angle of about 38 degrees and feed the rock to the crushers.

The main storage bins at the older rock houses are of the rectangular pattern, with sloping bottoms. The rock is drawn from these bins by means of apron chutes, operated by long levers. Of late years, most of the rock houses built are provided with large circular steel tank bins, of great capacity. The loading chutes in these circular bins are set in the bottom directly over the center of the railroad track. They are an improved pattern closed by small doors, operated by a system of levers from one central point. These new style chutes are more easily operated than the old aprons, and will probably replace the latter at our older rock houses.

Let us now consider what has been gained by these

changes: First, the large dump bin obviates the necessity of stopping the hoist in case of an ordinary delay at the crusher, such as the stopping of the crusher by a mass, etc. Second, we do away with screen bars and the expense of their upkeep. Under the old system Sunday repair work was a big item, since introducing the direct feed scheme, Sunday work has been almost entirely done away with. The sliding doors and chutes last from two and one-half to three years, or longer, as do also the cradle gates at the crusher.

Perhaps the biggest saving however, is in labor. Formerly a rock house crew consisted of six to eight men. At present two men will handle easily all that is hoisted.

Handling of mass copper and large boulders has been made comparatively easy by means of air lifts, trolleys and cranes, and whereas in olden times a rock house was about the hardest job around the mine, it has now become one of the easiest. Finally, to sum up the advantages in terms of actual saving in cost, the comparison of rock house expense before and since remodeling is as follows:

			Cost per ton	
Year.	Tons Crushed.	Labor.	Supplies, etc.	Total.
1906......	671,785	$0.0629	$0.0342	$0.0971
1911......	734,392	0.0262	0.0283	0.0545

PAST OFFICERS.

PRESIDENTS.

Nelson P. Hulst	1893	George H. Abeel	1903
J. Parke Channing	1894	O. C. Davidson	1904
John Duncan	1895	James MacNaughton	1905
William G. Mather	1896	Thomas F. Cole	1906
William Kelly	1898	Murray M. Duncan	1908
Graham Pope	1900	D. E. Sutherland	1909
W. J. Olcott	1901	William J. Richards	1910
Walter Fitch	1902	F. W. Denton	1911

(No meetings were held in 1897, 1899 and 1907.)

VICE PRESIDENTS.

	1893.	
John T. Jones F. P. Mills.	J. Parke Channing.	Graham Pope. M. W. Burt.
	1894.	
John T. Jones. F. P. Mills.	R. A. Parker.	Graham Pope. W. J. Olcott.
	1895.	
F. McM. Stanton. Geo. A. Newett.	R. A. Parker.	Per Larsson. W. J. Olcott.
	1896.	
F. McM. Stanton. Geo. A. Newett.	J. F. Armstrong.	Per Larsson. Geo. H. Abeel.
	1898.	
E. F. Brown James B. Cooper	Ed. Ball.	Walter Fitch Geo. H. Abeel.
	1900.	
O. C. Davidson. T. F. Cole.	M. M. Duncan.	J. H. McLean. F. W. Denton.
	1901.	
J. H. McLean. M. M. Duncan.	Nelson P. Hulst.	F. W. Denton. William Kelly.
	1902.	
William Kelly. Nelson P. Hulst.	Fred Smith.	H. F. Ellard. Wm. H. Johnston.
	1903.	
H. F. Ellard. Fred Smith.	James B. Cooper.	Wm. H. Johnston. John H. McLean.

	1904.	
H. F. Ellard. Wm. H. Johnston	Fred Smith.	John H. McLean. James B. Cooper.
	1905.	
M. M. Duncan. Fred M. Prescott.	F. W. McNair.	J. H. McLean. J. B. Cooper.
	1906.	
M. M. Duncan. J. M. Longyear.	Fred M. Prescott.	F. W. McNair. F. W. Denton
	1908.	
J. M. Longyear. F. W. Denton.	David T. Morgan.	D. E. Sutherland. Norman W. Haire.
	1909.	
W. J. Richards. Charles Trezona	D. T. Morgan.	D. E. Sutherland. Norman W. Haire.
	1910.	
W. J. Richards. John M. Bush.	Frederick W. Sperr.	Charles Trezona. James H. Rough.
	1911.	
E. D. Brigham John M. Bush.	Frederick W. Sperr.	C. H Munger. James H. Rough.

MANAGERS.

	1893.	
John Duncan Walter Fitch.	William Kelly	James MacNaughton. Charles Munger.
	1894.	
Walter Fitch. John Duncan	M. E. Wadsworth.	C. M. Boss. O. C. Davidson.
	1895.	
F. P. Mills. Ed. Ball.	M. E. Wadsworth	C. M Boss. O. C. Davidson.
	1896.	
F. P. Mills. Ed. Ball.	C. H. Munger.	Graham Pope. William Kelly.
	1898.	
M. M. Duncan. J. D. Gilchrist.	T. F. Cole.	Graham Pope. O. C. Davidson.
	1900.	
E. F. Brown. Ed. Ball.	James B. Cooper.	Walter Fitch. George H. Abeel.
	1901.	
James B. Cooper. James MacNaughton	(One Vacancy)	James Clancey. J. L. Greatsinger.
	1902.	
James Clancey. J. L. Greatsinger.	Amos Shephard.	Graham Pope. T. F. Cole.

	1903.	
Graham Pope.		T. F. Cole.
Amos Shephard.	Wm. J. Richards.	John McDowell.
	1904.	
John McDowell.		Thomas F. Cole.
Wm. J. Richards.	Graham Pope.	Amos Shepard.
	1905.	
John C. Greenway.		H. B. Sturtevant.
John McDowell.	William Kelly.	Wm. J. Richards.
	1906.	
John C. Greenway.		H. B. Sturtevant
Jas. R. Thompson.	William Kelly.	Felix A. Vogel.
	1908.	
James R. Thompson.		J. Ward Amberg.
Felix A. Vogel.	John C. Greenway.	Pentecost Mitchell.
	1909.	
F. E. Keese.		J. Ward Amberg.
W. J. Uren.	L. M. Hardenburg.	Pentecost Mitchell.
	1910.	
Frank E. Keese.		L. M. Hardenburg.
Charles E. Lawrence.	William J. Uren.	William J. West.
	1911.	
Charles E. Lawrence.		William J. West.
Peter W. Pascoe.	J. B. Cooper.	L. C. Brewer.

TREASURERS.

C. M. Boss 1893
A. C. Lane 1894
Geo. D. Swift 1895-1896
A. J. Yungbluth 1898-1900
Geo. H. Abeel 1901-1902
E. W. Hopkins ....1903-....

SECRETARIES.

F. W. Denton 1893-1896
F. W. Denton and F. W. Sperr....................1898
F. W. Sperr 1900
A. J. Yungbluth ....1901-....

LIST OF PUBLICATIONS RECEIVED BY THE INSTITUTE

American Institute of Mining Engineers, 99 John Street, New York City.

Mining and Metallurgical Society of America, 505 Pearl Street, New York.

American Society of Civil Engineers, 220 West 57th Street, New York City.

Massachusetts Institute of Technology, Boston, Mass.

Western Society of Engineers, 1734-41 Monadnock Block, Chicago.

The Mining Society of Nova Scotia, Halifax, N. S.

Canadian Mining Institute, Ottawa.

Canadian Society of Civil Engineers, Montreal.

Institute of Mining Engineers, Neville Hall, Newcastle-Upon-Tyne, England.

North of England Institute of Mining and Mechanical Engineers, Newcastle-Upon-Tyne, England.

Chemical, Metallurgical and Mining Society of South Africa, Johannesburg, S. A.

American Mining Congress, 1510 Court Place, Denver, Colo.

State Bureau of Mines, Colorado, Denver, Colo.

Reports of the United States Geological Survey, Washington, D. C.

Geological Survey of Ohio State University, Columbus, O.

Oklahoma Geological Survey, Norman, Okla.

Mines and Mining, 1824 Curtis Street, Denver, Colo.

Engineering-Contracting, 355 Dearborn Street, Chicago, Ills.

Mining & Engineering World, Monadnock Block, Chicago, Ills.

Mining Science, Denver Colo.

Mining & Scientific Press, 667 Howard Street, San Francisco, Cal.

University of Oregon, Library, Eugene, Oregon.

Case School of Applied Science, Department of Mining & Metallurgy, Cleveland, Ohio.

University of Illinois, Exchange Department, Urbana, Ills.

University of Missouri, Columbia, Mo.

University of Michigan, Ann Arbor, Mich.

Iowa State College, Ames, Iowa.

The Mexican Mining Journal, Mexico City, Mexico.

Geological Survey of New South Wales, Sydney, N. S. W.

Stahl und Eisen, Dusseldorf, Germany, Jacobistrasse 5.

LAKE SUPERIOR IRON ORE SHIPMENTS FROM THE DIFFERENT RANGES FOR YEARS PRIOR TO 1908
1908, 1909, 1910 and 1911, AND GRAND TOTAL FROM 1855 to 1911, INCLUSIVE.

(Compiled from Report Published by Iron Trade Review.)

		Prior to 1908	1908	1909	1910	1911	Grand Tot.
Marquette Range	(Tons....	85,233,187	2,414,632	4,256,172,	4,392,726	2,833,116	99,129,833
	(Per cent	22.2	9.3	10.0	10.1	8.6	18.8
Menominee Range	(Tons....	63,658,514	2,679,156	4,875,385	4,237,738	3,911,174	79,361,967
	(Per cent	16.7	10.3	11.4	9.9	11.9	15.1
Vermillion Range	(Tons....	27,175,626	841,544	1,108,215	1,203,177	1,088,930	31,417,492
	(Per cent	7.10	3.2	2.6	2.6	3.4	6.
Gogebic Range	(Tons....	54,032,590	2,699,856	4,088,057	4,315,314	2,603,318	67,739,135
	(Per cent	14.4	10.4	9.5	10.0	8.0	12.95
Mesabi Range	(Tons....	150,290,993	17,257,350	28,176,281	29,201,760	22,093,532	247,019,916
	(Per cent	39.4	66.3	66.1	67.2	67.4	46.9
Cuyuna Range	(Tons....					147,431	147,431
	(Per cent					.4	.0
Miscellaneous	(Tons....	675,419	122,449	82,759	91,682	115,629	1,087,938
	(Per cent	.2	.5	.2	.2	.2	.25
Total tons........		381,066,329	26,014,987 Decrease from 1907 38.5%	42,586,869 Incerase over 1908 63.6%	43,442,397 Increase over 1909 2.0%	32,793,130 Decrease from 1910 24.5%	525,903,712